Metallurgical and Chemical Engineering

A Consolidation of Electrochemical and Metallurgical Industry and Iron and Steel Magazine

Vol. VIII. New York, August, 1910 No. 8

PUBLISHED MONTHLY BY THE

Electrochemical Publishing Company

239 West 39th Street, New York.

Pittsburgh Office, 201 Oliver Building.
Western Office, 1510 Court Place, Denver, Colo.
European Office, Hastings House, Norfolk St., Strand, London, Eng.

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Yearly subscription price for United States, Mexico and United States dependencies, $2.00; for all other countries, $2.50 (European exchange, 10 shillings, 10 marks, 12.50 francs).

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Entered as Second-Class Matter at the Post Office at New York, N. Y., under the Act of Congress, March 3, 1879

New York, August, 1910.

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History Made by the Internal Combustion Engine.

It is interesting and not uninstructive to pause for a moment and reflect how the internal-combustion engine is making history. We trust it will not offend the friends of the "steamer" to suggest that it has practically made the automobile with its important and far-reaching effects of improving the health, if not the exchequers, of hundreds of thousands of people, giving employment to tens of thousands of men, circulating millions of dollars and making an irresistible sentiment in favor of rapid improvement of the public roads. It has made the motor truck, the gasoline car for light service of steam roads, and if it has not given us the flying machine, it has at least hastened its advent. All these things make history.

* * *

That is at one end. At the other end we have the very large gas engine, using blast-furnace or by-product coke-oven gas, promising to effect a complete revolution in these branches of the iron industry. The time was that the by-product coking process was looked at askance because its general adoption might glut the market for by-products. Not so when the large gas engine is at hand, for with electricity as a means of distribution the power market can never be glutten. By a curious coincidence, the internal-combustion engine makes another demand for a retort by-product, since it is obvious that the automobiles are destroying the old-fashioned macadam roads, while we are assured that tar will save them.

Monel Metal and Nickel Steel.

The treatment of the nickel-copper ores of the Sudbury district in Canada has long been a favorite subject of metallurgical research and invention. The classical process includes blast-furnace smelting and converter treatment near the mines at the works of the Canadian Copper Company, the 80 per cent nickel-copper matte from the converters being then shipped to the works of the Orford Copper Company at Bayonne, N. J., where separation of the nickel from the copper is obtained by means of the Orford process, which is well known in its general principles (smelting with salt cake and formation of tops and bottoms), but carefully guarded by secrecy with respect to some details. Aside from this orthodox treatment, quite a number of inventors have attempted the use of different methods, essentially wet processes with electrolysis. One of these, at least, has been worked on a commercial scale with commercial success for a number of years, that of Mr. David H. Browne, chief metallurgist of the Canadian Copper Company. His process has been described in detail in former volumes of this journal and will always remain a very instructive school example of an interesting combination process, involving chloride electrolysis and almost cyclic with respect to chlorine. When the Canadian Copper Company was absorbed by the International Nickel Company, the Browne process was given

up in 1902 in favor of the Orford process, because the latter is cheaper. One of the chief advantages of the Browne process, that it yields a much purer nickel than the Orford process, was considered not to be of sufficient commercial importance, in view of the limited market for highly refined nickel. The simplicity and effectiveness of the Orford process were the decisive moments.

* * *

A further step in exactly the same direction—toward simplification and cheapening of the metallurgical treatment—has been made in recent years by the introduction of "Monel metal." This is an alloy primarily of nickel and copper in the proportion of three parts of nickel to one part of copper, this being the natural proportion in which these metals exist in the largest deposits of nickel ore in the Sudbury district. The essential simplifying principle which considerably reduces the cost of the metallurgical treatment is that the nickel and copper are reduced together from the ore and directly obtained in form of the desired alloy and that no separation of the nickel from the copper is now attempted. The rationale of this simplified process is to be found in the remarkable physical properties of Monel metal, which were very well summed up in an article by Mr. D. H. Browne in our March issue, 1909 (Vol. VII, p. 114). Monel metal probably attracted first considerable general attention by the large extent to which it has been employed in sheet form to roof the new Pennsylvania terminal station in New York City. If the incorrodibility of this alloy should prove as great in extended practical use as it has been found in tests on a smaller scale, it should find many useful applications in the chemical industries in form of wire, sheet, or castings. At any rate, it is seen that the object of metallurgical research has been shifted by this development. The metallurgical treatment from ore to metal has been so simplified that little remains to be desired in this direction. The object must now be to so thoroughly investigate all the properties of Monel metal as to determine exactly its industrial possibilities and limitations, its commercial applications, in short to create a market.

* * *

In this respect the article in this issue by Prof. C. F. Burgess and Mr. J. Aston on the strength of the alloys of electrolytic iron and Monel metal is very interesting and suggestive. One of the chief uses of nickel is for making nickel steel. Now, Professor Burgess' investigation has shown that the addition of copper as well as of nickel in certain proportions to pure electrolytic iron improves its mechanical properties; further, that the two effects are additive, and that the desired improvement can be obtained by directly using Monel metal instead of copper and nickel. The commercial importance of this discovery is due to the fact that Monel metal is cheaper than metallic nickel. Hence, this result points toward a possibility of reducing the cost of nickel steel. However, the investigation has so far been carried out only with pure electrolytic iron, free from carbon. How the presence of carbon modifies the results must be left to further investigation, which is now being made by the authors. Of course, from a practical standpoint this is all-important. Yet, while not directly applicable to steel practice, the results obtained by Messrs. Burgess and Aston with pure electrolytic iron are important and valuable; the method of beginning their researches with pure electrolytic iron and then studying the modifications of the effects due to other additions is exactly logical and in strict agreement with one of the general rules so strongly emphasized by Dr. McKenna in his admirable presidential address before the American Institute of Chemical Engineers, printed elsewhere in this issue, on the study of the properties of materials—namely, that the simple or homogeneous substances must be the basis of study before the nature of the complex or heterogeneous can be understood. It is necessary to study first the effect of copper, nickel, and Monel metal on pure iron, before the effects of carbon and other elements in commercial steel can be taken up. The results given in the present article of Messrs. Burgess and Aston are certainly among the most interesting and suggestive so far obtained in their extended and painstaking research.

The Technical Graduate in Western Metallurgy.

Occasionally one hears a solicitous inquiry as to the disposition of the rapidly increasing number of technical graduates, particularly those from schools of mining and metallurgy. It is pointed out that the consulting field is already well filled, and that many large corporations place their work in charge of an engineering department with one or two efficient heads, and rely less and less on work done for them by independent engineers. Thus the prospects of the metallurgical engineer would seem to narrow down. But the problem is not so difficult as it may sound to some, and like many other problems which arise from changed conditions, it will find its solution in other changing ideas and practices. A decade ago Western smelters, then operating independently, found that efficiency in operation could be raised by the employment of technically trained men in what might be called the minor positions. Where formerly it had been thought sufficient to train a bright boy as assistant chemist and later promote him to take charge of the laboratory, it is now a settled rule that none but technical graduates are accepted in such positions. The result has been that through this system of successive promotions the Western smelting industry to-day finds itself directed in all of its important departments by young technical men who entered the work in minor positions.

* * *

The instance is but typical of what is happening in other branches of engineering. Large companies appreciate in the technical graduate not only the possibility of performing routine work more efficiently, but the probability of being able to advance him to places of greater responsibility which could not be held by the average so-called practical man. Thus we find that the "mill man" who has grown up in the service and undoubtedly knows his business well and has filled with a large degree of success his place in the development of metallurgical art, is in a measure now being displaced by the technical graduate who has enjoyed a broader training, is able to pay more attention to the refinements of the process, is willing to consider in an unprejudiced manner any proposals of modification and improvements and can undertake investigation and experimental work in an intelligent manner. The day of the average so-called practical man in many lines of metallurgy is passed; exceptions there are and will always be, but they only confirm the rule that in general he is superseded by the tech-

nical graduate. The arrangement makes for general improvement. Processes are developed and improved, routine work is handled with some degree of interest, the young graduate lays the foundation for a consulting practice later in life. Undoubtedly this system of using technical graduates will continue to spread until it will be no uncommon thing to find a corps of such men in charge of the actual work of our plants. The technically trained man cannot become a drug on the market if he understands the situation and accepts his place.

Railroad and Other Iron Consumption.

In our issue for October, 1908, under the caption "The Future of Iron and Steel Demand," we reverted to the cyclical nature of many of the occurrences in the iron industry, and also cited figures to indicate that railroad expenditures have also tended to move in a cycle. The intervening period has brought such confirmation of the views then expressed as can be furnished in so short a time. Without attempting to pass over the full ground then covered, it may be observed that we compiled a sort of "index number" to represent the increase in number of miles of road, of cars and of locomotives, weighting the items as seemed appropriate, thus deriving the following:

Annual average increase.	Index number.
1879-1887	129,921
1888-1893	107,209
1894-1898	26,656
1899-1907	180,424

Since 1907 railroad expenditures have been light, much as they were during the period following 1887. In the earlier period, or from 1889 to 1892, inclusive, the total production of iron and steel was large, far exceeding the output of previous years, and yet this large increase was attained despite a material decrease in railroad purchases.

* * *

Precisely the same thing has occurred since our former remarks were printed. Railroad buying has been very light, compared with the active period of 1905-6-7, but total demand has been heavier. In 1909 we made 25,795,471 tons of pig iron, against 25,307,191 tons in 1906 and 25,781,361 tons in 1907, but only 3,000,000 tons of rails against 4,000,000 tons in 1906 and 3,600,000 tons in 1907. Car building showed a much greater decline in 1909. In the first half of this year we made pig iron at the rate of 30,000,000 tons, a gain of more than 4,000,000 tons over 1906 and 1907, but the production of railroad material clearly showed a reduction. Were it possible to deduct the precise tonnage of material, each year, which goes ultimately into railroad consumption the increase in other lines of iron and steel consumption from 1906-7 to the present date would be shown to be tremendous. It is hard to account for the increase. The producers themselves can trace the material only occasionally. Most of the old lines of consumption have shown a growth commensurate with the increase in population and business activity, but, as a rule, this is quite insufficient to explain the total increase, for general business does not seem as good as it was three and four years ago.

Crushing and Grinding.

Radical change and progress are imminent in the matter of crushing and grinding ores. In the 55 years from 1835 to 1890 stamps increased in weight from 50 lb. to about 1000 lb., at which weight they have largely remained in general practice in the United States and Australia. In the last decade, however, the tendency in South Africa has been toward heavier stamps, and the weight of the individual stamp has been increased from about 1000 lb. to 2000 lb. The change has been accompanied by the adoption of two-stage crushing, using the heavy stamp as a coarse crusher, and a regrinding mill, usually the tube mill, as the means of reducing the ore to its final size. The light stamp, crushing the ore to the desired degree of fineness in one operation, has been relegated to a minor place in South African practice. The development of heavy gravitation stamps has been detailed by Dr. W. A. Caldecott in a communication to the Institution of Mining and Metallurgy, and his paper has caused a widespread and interesting discussion among engineers in the Institution and in the South African society. The paper by Dr. Caldecott, together with part of the discussion, will be found elsewhere in this issue.

* * *

The discussion has brought out one point very clearly, viz., while it is true that the heavy stamp will have a higher duty than the light stamp, it does not at once follow that this makes for greater efficiency, in view of the necessity of some sort of regrinding apparatus. In other words, the relative economic efficiency of single-stage and two-stage crushing is not proved by the mere fact that heavy stamps have a high duty. As to the relative efficiency of stamps and tube mills, it has been claimed that if the mesh to which the ore is to be crushed be properly adjusted to the weight of the stamp, the stamp mill will show probably greater efficiency than the tube mill, and this has led one of the opponents of two-stage crushing with stamps and tube mills to argue that if two-stage crushing is necessary in order to cope with larger tonnages of ore, it might be found advisable to use stamps for both the coarse and fine crushing, sizing the product of the heavy stamps before treatment in the light. But this argument overlooks one general metallurgical rule, namely, that when, as is so often the case, a metallurgical process is resolved into two consecutive stages or operations, essentially different machinery are employed in both stages, such are are most suitable to the characteristic features of each stage.

* * *

Dr. Caldecott is of the opinion that the present use of heavy stamps makes further investigation of light stamps largely academic, but this view cannot be shared until it is shown by sufficiently extended experience that the heavy stamp tube mill combination is more efficient from the financial point of view than single-stage crushing. Probably this will turn out so. Again, other regrinding devices have not enjoyed the popularity of the tube mill, and it is not impossible that at least under certain conditions one of them might be used in place of the tube mill if two-stage crushing is generally adopted. But this is simply hypothetical. Evidently a subject has been opened which will not be closed to discussion and experimentation for some time. The tests have brought out a great deal of useful knowledge, but they have revealed some weak points which further experiment must strengthen if the advocates of two-stage crushing expect to maintain their present strong position.

Chicago Meeting of the American Electrochemical Society.

A new section of the American Electrochemical Society was organized on July 7th in Chicago. Prof. Herbert M. McCoy, of the University of Chicago, has been elected chairman, and Mr. A. B. Marvin secretary-treasurer of the section. The first business transmitted by the Chicago section has been to extend to the national society a hearty invitation to meet in Chicago in October.

The society has accepted with great pleasure the invitation of the Chicago section, and as the Wisconsin section will co-operate, a highly interesting and profitable meeting is assured.

Among the visits which are to be arranged is one to the electric steel refining plant (Heroult furnace) of the Illinois Steel Company (U. S. Steel Corporation) at South Chicago, and another one to the electrolytic lead refinery (Betts process) of the U. S. Smelting, Refining & Mining Company at Grasselli. Possibly a trip on the way back east to Sault Ste. Marie will be arranged. The date of the meeting will be from Oct. 13 to 15.

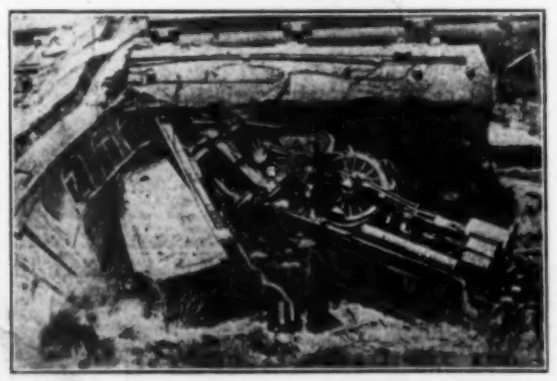
The Wreck of the Special Train of the American Chemical Society.

As noticed before in these columns, a special train had been arranged to take the Eastern members of the American Chemical Society to the San Francisco meeting. The train left Chicago on July 4 and was to arrive in San Francisco on July 12. On the latter date we received, through the courtesy of Dr. L. H. Baekeland, the following telegram:

"Our chemical special train fearful smash-up near King City. Miraculous escape of all passengers. Locomotive turned turtle, sleeping car jumping over it in air, ditching right and left. Everybody calm and self-possessed. Engineer, stoker and conductor killed. Party arrived here by next train. Otherwise trip was most splendid."

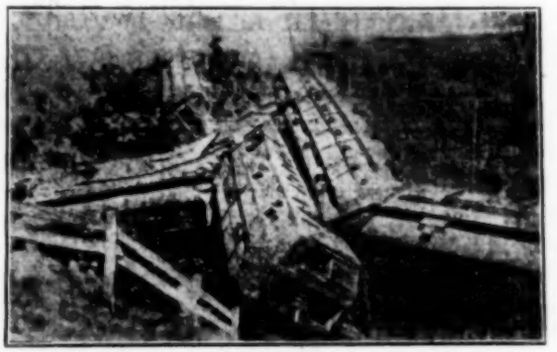
How serious the accident was may be seen from the adjoining illustrations, being reproductions from the San Francisco *Call*, the first being a photograph made by a reporter of that paper, the second a snapshot taken by Mr. J. B. Thomas, a passenger on the train.

Excessive speed in a restricted district is said to have been responsible for the wreck. Among the members of the American Chemical Society who were on the train were Mr. and Mrs.



WRECK OF TRAIN OF CHEMICAL SOCIETY.

George P. Adamson, with daughter, of Easton, Pa., Prof. and Mrs. W. D. Bancroft of Cornell, Dr. and Mrs. L. H. Baekeland of Yonkers, N. Y., Mr. and Mrs. John T. Baker of Phillipsburg, N. J., Dr. and Mrs. H. E. Barnard of Indianapolis, Ind., Mr. W. Beardwood, Dr. Walker Bowman of New York, Mr. and Mrs. H. B. Bishop, Mr. and Mrs. I. M. Bregowski of Chicago, Dr. H. H. Bunzel, Dr. and Mrs. Eug. A. Byrnes of Washington, D. C., Dr. Frank D. Cameron of Washington, Mr. W. E. Chamberlain, Mr. A. N. Clark of Berlin, Ontario, Miss Edna A. Clark of Washington, Mr. and Mrs. A. M. Comey of Chester, Pa., Mr. and Mrs. C. F. Crowley of Omaha, Neb., Mr. E. de Meeus, Prof. W. L. Dudley of Nashville, Tenn., Mr. L. Feldstein, Mr. Albert Finke of Cleveland, Ohio, Prof. G. B. Frankforter of University of Minnesota, Mr. L. E. Gallagher of Arlington, Mass., Prof. M. Gomberg of University of Michigan, Dr. A. P. Hallock of New York, Mr. J. F. Harbow, Mr. F. W. Hatchel, Mr. and Mrs. Elwood Haynes of



WRECK OF TRAIN OF CHEMICAL SOCIETY.

Kokomo, Ind., Mr. Morris W. Hedden of Boston, Mass., Mr. W. F. Health of Boston, Mass., Dr. C. H. Herty of Chapel Hill, N. C., Dr. W. F. Hillebrand of Washington, D. C., Prof. J. I. D. Hinds of University of Nashville, Miss Hines, Mr. W. C. Hovey of Chicago, Ill., Mr. George M. Howard of Philadelphia, Pa., Dr. and Mrs. Treat B. Johnson, of New Haven, Conn., Mr. W. M. Kelsey of Depue, Ill., Prof. and Mrs. E. F. Ladd of Agricultural College of North Dakota, Mr. and Mrs. Charles Lamont of Detroit, Mich., Miss Mary F. Leach of Western College, Oxford, Ohio, Dr. M. Le Clark of Washington, Dr. E. G. Love of New York, Prof. H. McCormack of Armour Institute, Chicago, Prof. W. A. Noyes of University of Illinois, Mr. and Mrs. M. F. Nichols, Mr. J. F. Oakleaf, Prof. and Mrs. Charles L. Parsons of New Hampshire College, Durham, N. H., Mr. A. M. Patterson of Columbus, Ohio, Mr. and Mrs. F. B. Porter of Atlanta, Ga., Miss A. P. Schaffner, Mr. R. N. Shreve of St. Louis, Mo., Prof. C. F. Sidener of University of Minnesota, Mr. and Mrs. Frank N. Smalley, Mr. and Mrs. H. R. Smith of Highland Park, Ill., Mr. M. O. Stafford of Philadelphia, Dr. T. C. Stearns of Jersey City, N. J., Mr. F. L. Steenken, of Brooklyn, N. Y., Mr. C. J. Strosacker of Midland, Mich., Mr. and Mrs. Edw. R. Taylor of Penn Yan, N. Y., Dr. Charles E. Taylor, Misses Clara E. and Alice E. Taylor, Mr. J. B. Thomas of Baltimore, Md., Dr. Willis G. Tucker of Albany, N.Y., Prof. and Mrs. George F. Weida of Kenyon College, Gambier, Ohio, Dr. W. R. Whitney of Schenectady, N. Y., and Mr. E. Wolensky of Crete, Neb.

The only members of the party of chemists injured were in the car just back of the dining car. None was hurt seriously enough to be taken to a hospital and they resumed the trip with their friends.

According to the reports received the miraculous escape of the passengers was due to all-steel cars.

We reserve for our next issue a report of the San Francisco meeting itself, which turned out exceedingly successful.

The Iron and Steel Market.

The closing of steel works in July for repairs was inconsequential, a few plants closing for from 10 days to three weeks, while many did not close at all. Production, therefore suffered very little. There has only been a slight further decrease in pig iron output, which at the close of July is probably but little under 27,000,000 tons a year, representing a total reduction of about 15 per cent as compared with the maximum

rate, reached in February, and easily exceeding the average rate in 1906 and 1907.

The general tendency of buyers is to make purchases sparingly and to reduce stocks, the condition being in striking contrast to that of the fall and winter, when specifications were liberal and material was being accumulated. With this complete reversal in the attitude of buyers the flow of material is well maintained, not having decreased more than about 15 per cent at the outside. Thus the evidence is that the ultimate consumption is in excess of the current production, and is decidedly larger than it was six or nine months ago.

Indeed, the persistence of a heavy tonnage demand is remarkable, particularly when it forms such a striking contrast to the general apathy observed in market circles. Superficially, the market is intensely dull throughout; practically, there is a tremendous amount of material being made, shipped and consumed.

The trend of prices continues downward, although very moderately. Pig iron has been almost stationary in the past month, while prices of finished steel products have yielded only at a few points, and then only by a small amount.

On July 14 the American Steel & Wire Company began quoting plain wire at $1.50, base; wire nails at $1.70, base; painted barb wire at $1.70, and galvanized barb wire at $2. These prices are $3 per ton below the prices as advanced Dec. 13 last, but the actual reduction at the moment in open-market prices was only $1 a ton. In February shading of $1 a ton appeared, and this was increased to $2 about the beginning of July, this latter shading being only by the independents, the leading interest finally recognizing the condition and cutting an extra dollar so as to establish a market.

The shading in sheets has increased from $1 to $2 a ton since last report, and now amounts, in extreme cases, to $3 a ton on black and $4 a ton on galvanized products.

In plates, shapes and bars prices are unchanged, but the expectation is for lower rather than higher prices. Merchant steel pipe is strictly held, but this is due to demand being slack rather than to its being heavy, for the jobbers still have large stocks and the mills wish to protect them, realizing also that cut prices at this time would not bring out more business.

On July 19 an agreement was reached between the Republic Iron & Steel Company and the Amalgamated Association of Iron, Steel and Tin Workers relative to wage scale for the year beginning July 1. The Republic company operates a number of iron mills, which are under Amalgamated control, while a few of its steel rolling mills are similarly controlled, its large steel plant and other operations being non-union. The scales adopted make an advance of 50 cents a ton in puddling as long as bar iron is 1.25 cents per pound or higher, i.e., the base of the scale remains $5 per ton, but advances to $5.12½ on 1.05-cent bar iron instead of on 1.25-cent bar iron, subsequent advances being 12½ cents in puddling for each 1/20 cent in bar iron. The last bimonthly adjustment showed that the average realized price on base sizes of iron bars shipped from the selected Ohio mills in May and June was 1.46375 cents, which calls for $6.12½ to be paid in July and August, while the expiring scale would call for $5.62½. The settlement is a restoration of the 1907-8 puddling scale. The other iron mills scales were advanced an average of about 4 per cent, falling short of restoring the 1907-8 scales. The Western Bar Iron Association, composed chiefly of independent mills in the West, has accepted the Republic settlement.

Pig Iron.

Apparently pig iron prices have closely approached bottom in the sharp decline which has been in progress since last November, for the settling since last report has been very slight. Valley and Birmingham prices, indeed, are not quotably lower, but there have been slight declines along the Lakes and in the East. The market has been extremely quiet. Consumers are buying only for immediate requirements and are curtailing stocks in many cases. The merchant furnace production has been steadily reduced, but is still rather heavy in comparison with actual market demand. At the close of July, 11 of the 21 merchant furnaces in the Mahoning and Shenango valleys are in blast, and some iron is probably being piled. The time was that "the valleys" occupied a strategic position of great value, being able to distribute iron in all directions, but furnaces have grown up around them in a circle, greatly restricting their territory, and since the abrupt cessation of buying of Bessemer iron by the large steel works, late last year, the valley position has been poor. While $11.50, Birmingham, is quotable as the market for delivery through the second half there is some shading from this level. Valley prices remain quotable as follows, freight to Pittsburgh being 90 cents: Bessemer, $15.50; basic, $14.50; No. 2 foundry, $14.50; forge, $13.75; malleable, $15.

Steel.

There has been a sharp decline in open-hearth billets, which have become available at $26, Pittsburgh, $1.50 under last quotation, while Bessemer billets are easy at $25, Pittsburgh, having been firm at $25 to $25.50 a month ago. There has been another curious reversal in the position of raw materials for steel making. In the past heavy melting steel scrap has regularly sold at less than basic pig iron, delivered at works in the Central West. At the beginning of 1908, on account of the sudden slowing down of the whole industry leaving a large tonnage of material in transit, prices were particularly low, and predictions were then freely made that open-hearth billets would ultimately sell at less than Bessemer. Last year, on the rapid increase in operations, there was developed a corresponding scarcity of scrap, and prices passed far above pig iron. The pendulum has swung again in the opposite direction and melting steel is more than a dollar a ton below basic pig iron, delivered in the Central West, or $2 a ton below Bessemer. The industry has not yet reached the point where cost of production dictates relative prices, but the trend is distinctly in the direction of establishing a cost-of-production relation between prices of Bessemer and open-hearth steel. We quote the market, Pittsburgh, as follows: Bessemer billets, $25; sheet bars, $26 to $26.50; open-hearth billets, $26; sheet bars, $27 to $27.50; rods, $29 to $29.50.

Finished Products.

Regular prices, f.o.b. Pittsburgh, are as follows, subject to shading of $2 to $3 on black sheets and $3 to $4 on galvanized:

Standard steel rails, $28 for Bessemer, $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.40 to $1.45 for tank quality, ¼-in. and heavier.

Shapes, $1.40 to $1.45 for beams and channels, 15 in. and under, zees and angles 2 in. x 3 in. and larger.

Steel bars, $1.45, base; hoops, $1.50 to $1.60; iron bars, $1.50, Pittsburgh; $1.40, Philadelphia; $1.40, Chicago.

Wire nails, $1.70, base; plain wire, $1.50, base; painted barb wire, $1.70; galvanized barb wire, $2.

Black sheets, 28-gage $2.40; galvanized $3.50; blue annealed, 10-gage, $1.75; painted corrugated roofing, 28-gage, $1.70 per square; galvanized, $3; tin plates, $3.60 for 100-lb. cokes.

The Non-Ferrous Metal Market.

The market has been practically without feature during the month. Slight fluctuations, either up or down, have not shown any permanency, and the prices have returned to a level from which it seems impossible to move them.

Copper.—A rather large volume of business has been transacted owing to the fact that manufacturers have remained out of the market so long that their stocks were low. Considerable activity has been noticeable in electrolytic, especially for August and September delivery. Lake copper has been quoted at 12⅝ cents to 12¾ cents, with electrolytic at 12⅛ cents to 12⅜ cents.

Tin.—Only such business as is imperative is being transacted. Few purchasers for future delivery are being made and most manufacturers are inclined to buy for immediate needs only.

Lead.—The market is rather quiet, but prices have ruled firm at about 4.25 cents to 4.27 cents at St. Louis, and 4.35 cents to 4.40 cents at New York. Buying has been without feature, and business is nominal from day to day.

Zinc.—Buying has been sparing on the part of manufacturers in the galvanizing business and as a consequence the market has been very quiet. Prices have eased off to 5 cents and 5.05 cents, St. Louis, and 5.15 cents to 5.20 cents, New York.

Other Metals.—A comparatively small volume of business is being transacted in aluminium, antimony, mercury, etc., but the prices remain unchanged. Aluminium is quoted at 23¼ cents per pound, New York; antimony, 8.15 cents to 8.20 cents per pound for Cookson's; mercury, $47 to $49 per flask of 75 lb., according to quantity taken.

The Western Metallurgical Field.

Sampling Spelter.

The inaccuracy of the drilled sample as compared with the granulated sample in spelter analysis was clearly shown at a recent meeting of the Colorado Section of the Western Association of Technical Chemists and Metallurgists, when Mr. E. M. Johnson presented some results that had come under his supervision. Both drilled and granulated samples from the same lot will show a variation in the lead and iron content, but the variation will not be so great in the latter as in the former. For instance, in drilled samples from one lot the variation in iron was 0.008 per cent, while in granulated samples it was only 0.004 per cent. In another lot the variation in iron was 0.036 per cent in the drilled samples, while it was but 0.02 per cent for the granulated.

With regard to lead the difference is even more marked. In one lot the greatest difference in the drilled samples was 0.32 per cent, while with granulated samples it was but 0.09 per cent. In many other tests the same difference in favor of the granulated sample was noted. In addition to the fact that the granulated sample is more accurate, it has other advantages pointed out by Mr. Johnson. First, the metallurgist has a much better control over his spelter, as he can get results sooner. This may save rehandling the spelter, if immediate shipments can be made. Second, the cost of drilling samples amounts to 75 cents or $1 per day, which is eliminated with granulated samples, as they are usually taken by the furnace attendant or foreman. Another point in connection with granulated samples is that when properly taken they go into solution more readily than do the drilled samples.

Bureau of Mines.

Western interests have not been satisfied with the delay in appointing a permanent director of the Bureau of Mines lately created. There is a tendency to regard the appointment of George Otis Smith, present director of the Geological Survey, as temporary director of the bureau, as a mere postponement of an issue which has been evident from the beginning, viz., whether an ardent friend or an opponent of the bureau shall be its head. Unquestionably, if the bureau is to amount to anything and perform the duties for which it was created, it must be directed from the beginning by one who is friendly to its useful existence. The consensus of western opinion is that Joseph A. Holmes, former director of the technologic branch of the survey, is the logical man for the place and it is hoped that when the President finally makes his appointment Dr. Holmes will be found at the head of the new bureau.

American Mining Congress.

The official call for the thirteenth annual convention of this congress has been issued. The convention will be held at Los Angeles, Cal., from Sept. 26 to Oct. 1. The meetings will provide opportunity for the discussion of the production and treatment of useful minerals, conservation in its relation to the mining industry, uniform mining laws in the different States, and other questions that concern mining and metallurgical progress. A large attendance is expected and arrangements for a good meeting are now under way. The program will be announced later.

International Company's Tooele Smelter.

August will probably witness the blowing in of the new copper smelter at Tooele, Utah. Five reverberatory furnaces will constitute the initial smelting equipment of the plant, and these are rapidly nearing completion. The roaster equipment has been installed for some time and the balance of the plant has been tested so that everything is in readiness for operation. The principal ore supplies will come from the mines of the Utah Consolidated at Bingham and the South Utah, formerly the Newhouse Mines & Smelters Corporation, at Newhouse. The ore from the former will be delivered by a tramway recently completed by the Trenton Iron Company. The contract with the South Utah company is more recent and will provide for the shipment from Newhouse of about 200 tons of copper concentrates per day, beginning with the first of August. The Tooele plant will represent the highest state of the smelting art and its operation will be watched with considerable interest.

Colorado.

It is stated that in June, 1910, Colorado passed the billion-dollar mark in metal production. Be that as it may, it is certain that the industry in this State is not in as flourishing a condition as it should be, and steps have been taken to look into the causes of the decline. Colorado is not willing to relinquish her position as one of the leading mining States, despite the fact that her smelters and mills are either idle or run at reduced capacity. At a meeting of mining men held at Denver on July 14 to consider the mining and metallurgical situation three phases of the matter were discussed, viz., ore treatment and transportation rates, the deficiency of present metallurgical processes and the relations of mining and the forest service.

The rates on ore treatment and transportation were regarded as one of the causes of the decline of mining in the State. It was pointed out that ore in certain districts had so changed in its metal content that the smelting charges at present in force made it impossible to mine the ore at a profit. Freight rates were considered excessive, especially as compared with rates in other States where ore is hauled over mountain railroads with heavy grades. A reasonable reduction in both freight and smelting charges would provide mining expenses and permit much ore to be mined and treated that must now remain untouched.

The fact that progress in metallurgy has not kept up with the requirements of mining has been mentioned before, and is undoubtedly a valid reason for the present unsatisfactory state of mining, not only in Colorado, but in other Western States. There are many ores the gross metal of which is sufficiently high to warrant its treatment if all, or nearly all, of the values could be recovered, but in our present state of metallurgy a complex ore suffers losses in the way of by-products and treatment charges that make it unprofitable. It is firmly believed by many metallurgists that the process of the future will be chemical, and that it will recover practically all of the valuable constituents of the ore. Steps in this direction are already in evidence and the day of their success is awaited with interest.

The present relations between the Departments of the Interior and Agriculture, whereby prospecting on the public domain has been subject to unnecessary restrictions, were blamed in large measure for the failure to develop new mining districts. Formerly it was necessary only that a miner should perform his annual assessment work to secure title to a mining claim, but under the regulations of the forest service a prospector on the

forest reserves, which cover much of the most attractive ground in the State, must submit to the opinion of the ranger as to whether he has a valid and valuable mineral discovery. It is believed that these restrictions have been one of the reasons why no important new mining districts have been discovered and exploited in Colorado for many years. A change in the regulations is gradually being made, however, and it is believed that in a short time the regulations will not be heard of.

Partridge Hot Blast Furnace.

A demonstration furnace of this type has been established at the old Denver Paper Mills, near Denver, Colo. The Partridge furnace is the invention of A. R. Partridge, who has erected a number of plants in Mexico and a few in the United States. The main features of the furnace are a hearth, or crucible, which can be lowered from the shaft; an opening of about 3 in. or 4 in. between crucible and shaft when the former is in position, this opening serving in lieu of tuyers for the admission of the light blast, and finally a system of heating the blast with the waste heat from the slag by discharging the slag into an endless conveyor which travels in a steel housing through which the blast is directed. The air blast passes through this inclosure surrounding the slag conveyor and is finally delivered to a wind box surrounding the furnace between the bottom of the shaft and the crucible where it enters the former. The inventor claims for his furnace simplicity of design and operation, and adaptability to the treatment of lead or copper ores, producing either bullion or matte. The demonstration furnace at Denver is attracting some attention and is in operation almost daily on various classes of ore.

Goldfield Consolidated.

The June report of the Goldfield Consolidated Mines Company, Goldfield, Nev., shows some interesting metallurgical results. The mill treated 20,091 dry tons of ore of an average value of $45.27 per ton. The total value of the ore was $909,480; loss in tailings, $41,216; value received, $868,264; percentage extraction, 95.47. The extraction of the mill was unusually good, and in this connection it is reported that the tailings loss included more than 300 ounces of gold in concentrate residues. These residues are usually shipped to the smelters, but in June more than half the concentrate residues were too low-grade to stand the cost of shipment and were thrown on the dump. Practically all of the damage caused by the fire of last April has been repaired and the mill is nearly in condition to handle its full capacity of 860 tons daily. At present it would be impossible to increase the tonnage milled on account of the shortage of water, but this is expected to be but a temporary condition. In all other respects the plant is in good condition.

Corrosion of Non-Ferrous Metals.

The Council of the Institute of Metals in London have decided to undertake an investigation into the causes of the corrosion of non-ferrous metals by sea water, acids, etc., and by other chemical and electrolytic reactions.

A committee, known as the Corrosion Committee of the Institute of Metals, has been appointed for the purpose of carrying out the investigation.

The following are the names of the gentlemen constituting the committee:

Sir Gerard Muntz, Bart., chairman of the committee (president of the Institute of Metals).

Prof. H. C. H. Carpenter, of the University of Manchester.

Capt. G. G. Goodwin, deputy engineer-in-chief, Royal Navy.

Prof. A. K. Huntington, King's College, London.

J. T. Milton, chief engineer surveyor, Lloyd's Register of British Shipping.

A. Philip, Admiralty chemist.

Leonard Sumner, of the Broughton Copper Company, Ltd.

Prof. T. Turner, of the University of Birmingham.

Sir William H. White, past-president of the Institute of Metals.

The first meeting of the committee was held at the offices of the Institute, Caxton House, Westminster, S. W., on May 24, all the members being present with the exception of Sir William White, who had just left for a visit to the United States of America.

Realizing the magnitude of the task before them, and the vast number of ramifications to which such an investigation must lead, the committee have decided in the first instance to confine their attention to the question of the corrosion of condenser tubes in marine engines and in stationary engines using foul water or being subject to violent electrolytic action, such as often occur in electric power stations.

Before commencing experimental research the committee propose to have prepared an epitome of such information on the subject as is already available from previous research in the same direction. Mr. G. D. Bengough, of the Metallurgical Department of the University of Liverpool, has been asked to undertake this work and to become a member of the committee.

CORRESPONDENCE.

Rapid Electrodeposition.

To the Editor of the Metallurgical and Chemical Engineering:

Sir—I notice in your issue of May a communication by Allen P. Ford referring to my paper with Mr. Hughes, in which we describe some new forms of rotating electrodes for the quantative deposition of metals. Mr. Ford thinks it strange that in our paper we made no mention of the article by Dr. J. T. Stoddard in which he gives interesting information as to the merits of stationary wire gauze cathodes. The reason for not noticing the paper of Dr. Stoddard and also of other investigators along the same lines was because the paper was simply dealing with electrodes which had been gradually evolved in my laboratory.

I am myself by no means wedded to the use of rotating electrodes and am inclined to think that rather too much is made of rapid methods of deposition, although I know I have been rather a sinner in this way myself. I am not sure, but I rather believe, that I was the first to use rotating electrodes, my first one having been made in 1899, although I never published that particular form.

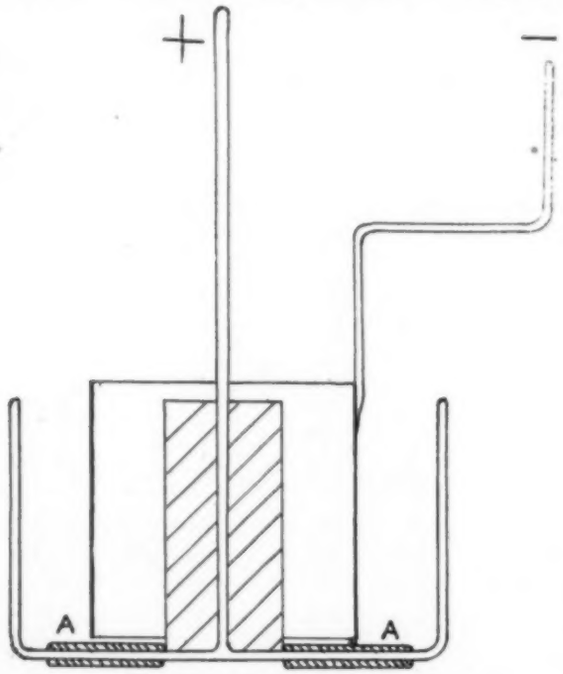
The motive power was clockwork and the rotation was four revolutions per minute. The anode was rotated and consisted of stout platinum wire (10 B. W. G.) which was bent as illustrated in the figure and having a piece of sheet platinum fastened on to the upright to cause a swirling of the electrode. The cathode consisted of the old style of cylindrical electrode. At *A A* the anode was insulated by means of pieces of glass in order to prevent uneven deposit on the edge of the cathode. In fact, it was because at that time I was using the cylindrical sheet cathode that this anode was devised so as to give even deposition inside and outside the electrode. By means of this electrode the time of deposition was more than halved and higher current densities could be employed without "burning." I believe that by using rapid rotation and substituting gauze for the plain cylinder that this apparatus would work very well. The chief difficulty is to obtain rigidity.

Very shortly after this I devised a rotating gauze cathode which was described before the first meeting of the Faraday Society. Since then I have experimented with a large number of different types, and the paper read before the Faraday Society in January was the outcome of this work.

At the same time, however, I have, for a great deal of my work, used a stationary gauze flag electrode, which I also described at the first meeting of the Faraday Society in June, 1903. I have at times used a gauze cylindrical cathode with a spiral platinum wire as anode, and I agree that very rapid deposition does take place, but I have not succeeded in deposit-

ing out the whole of the metal quite so rapidly as Mr. Ford has.

The drawback to very rapid methods is that they require so much attention. Personally, when possible, I much prefer to get a number of solutions ready and electrodes weighed, and



ROTATING ELECTRODE ARRANGEMENT

allow the electrolysis to take place over night. There are, of course, cases, notably that of bismuth, where it is by no means easy to obtain adherent and good deposits with stationary electrodes, and in such cases the rapid methods should be employed.

London, England. F. Mollwo Perkin.

* * *

Temperatures of Electric Steel Refining.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The chief advantage of the electric furnace over the open-hearth for steel refining is generally considered to be due to two causes—the higher temperature and the neutral atmosphere. However, with respect to the higher temperature this statement should be somewhat modified. It is true that it is quite possible to produce much higher temperatures in the electric steel furnace than in the open-hearth furnace, and as a matter of fact there exist locally much higher temperatures right below the arcs. In general, however, contrary to the usual assumption, it is a fact that the average temperature of the electric steel furnace is not much higher than that of other metallurgical furnaces.

I may first refer to your article in the April issue on the 15-ton Héroult furnace at the South Chicago Works of the U. S. Steel Corporation, giving some temperature measurements of Prof. Jos. W. Richards with the Mesure-Nouel pyrometer which are so low as to show that the electric steel was in the average not hotter than the steel in the converter. The reason for this might be found in the fact that in the South Chicago works the bessemer metal is refined for only one to two hours in the electric furnace and that only 200 kw-hours are supplied to the steel per ton, so that the refining process yields only a good average quality (0.4 C, 0.4 Mn, 0.03 S and 0.03 P), but I have made myself similar temperature measurements with a Wanner pyrometer in a 5½-ton arc furnace in which steel of the highest quality was produced, and have found almost exactly the same average figures of temperature as Professor Richards. The suggestion of Dr. Amberg in your June issue (page 314) that the results were doubtful because the Mesure-Nouel pyrometer was unreliable, is, therefore, disproved.

The furnace which I tested was a Nathusius furnace of 5½ tons capacity, which has been in operation on the Friedenshütte in Upper Silesia for a year.[1]

Converter steel was charged into the electric furnace and refined to a very soft steel containing 0.5 to 0.06 C and very small amounts of sulphur and phosphorus (0.003 to 0.02 S and 0.004 to 0.017 P). The treatment lasted for 3½ to 4½ hours and the energy consumption per ton of steel was 300 to 400 kw-hours.

The converter metal when poured into the smaller ladle had still a temperature of 1450° C. to 1470° C. When poured from the ladle into the electric furnace the temperature was 1435° C. to 1450° C. The interior of the furnace (immediately after the pouring of the finished steel) had a temperature of 1535° C. to 1560° C., the points of the electrodes, when being withdrawn, being at 1300° C. to 1350° C. The temperature of the electric steel when poured was 1500° C. to 1520° C. for a soft charge and 1558° C. for a high silicon charge. In the latter case the temperature when pouring into the ingot was still 1500° C.

Since the electric furnace had always to wait a certain time for a charge of fresh bessemer metal, the interior of the furnace cooled to about 1000° C. After the metal had been charged into the furnace and the reaction was completed, the temperature of the slag layer was measured and was found to be, after an hour, 1550° C., and after an hour and a half, 1600° C. to 1650° C. It remained at 1600° C. to 1650° C. until the slag was removed, when its temperature was found to be between 1600° C. to 1560° C. The removal of this first slag and the introduction of the dephosphorizing slag caused the bath to cool off somewhat to about 1550° C. and the temperature remained in the second period of the treatment a little less than in the first period.

Concerning the Roechling-Rodenhauser induction furnace I have no numerical data of the temperature, but skilled open-hearth furnace engineers estimated the average temperature not to be much higher than in the open-hearth furnace.

What do these figures prove? They show evidently that an abnormally high temperature is not necessary for the refining process. As a matter of fact, even the heat of the arc for the local heating of the slag is not absolutely necessary, since it is generally known that the Roechling-Rodenhauser furnace permits the same far-reaching refining reactions to be carried out down to traces of sulphur and phosphorus without any arc.

The refining possibilities of the electric steel furnace are, therefore, certainly due to a larger extent to the non-oxidizing atmosphere than to a very high temperature. There is no doubt that the possibility of producing a higher temperature in the electric furnace helps the refining process considerably, since the higher basic slag can be easily kept fluid, but as mentioned above the arc furnace is not absolutely necessary for this purpose.

On the other hand, it is quite self-evident that at the places right below the electrodes the temperatures are much higher than the above average temperatures. The proof of this is the formation of calcium carbide, which has often been observed in the arc furnace and which according to the latest investigations (Metallurgical and Chemical Engineering, 1910, pages 279 and 328) begins only at about 1820° C. and which does not occur in the induction furnace. Further, the electric steel furnace offers the possibility of heating the metallic bath for special purposes to a much higher temperature than is possible in other furnaces. I have myself seen steel tapped at temperatures above 1650° C., but in the average the mean temperature required for the almost complete removal of phosphorus and sulphur is not much higher than in other metallurgical furnaces.

Bernhard Neumann.

Institute of Technology,
Darmstadt, Germany.

[1] For further description of this furnace see the abstract of the paper of Prof. Neumann in the report of the International Congress of Mining and Metallurgy, printed elsewhere in this issue.—Ed.

Kr. Birkeland.

Kr. Birkeland, Norwegian physicist, was born in Christiania, in the year 1867, and is now professor of natural philosophy at the University of Christiania.

He received his first scientific education at this university, and later he studied under Poincaré, Picard, and Appel, in Paris, and under Hertz in Bonn.

The first work of his to attract notice concerned a complete solution of Maxwell's equations for the electromagnetic field; he had already carried out a series of investigations on electrical vibrations and on discharges through gases.

He was the first in Norway to take Röntgen photographs, and did so only a few weeks after the incomplete report of the X-rays was received, and with apparatus made by himself, which is a proof of his keen attention in all fields and of his wide scholarship. He then threw himself with great energy into the study of cathode rays and investigated in particular the deflexion of these rays under magnetic influence.

It is an indication of his scientific imagination that he was not satisfied with a detailed investigation of these rays, but soon perceived what immense importance they possessed in the energy of the universe. He especially directed his attention to the sun and in this connection enunciated his well-known theory of sun spots.

Kr. Birkeland

These investigations led him to a theory of the northern lights (aurora borealis) which he explained as produced by cathode rays or related rays emanating from the sun and absorbed toward the earth's magnetic poles.

It is an interesting trait of his scientific personality that he was not content with forming a general theory of northern lights, but submitted it to a searching test, both in the physical laboratory and by exact mathematical analysis; in the latter respect his colleague, Professor Störmer, has supported his theory of the northern lights in an effective manner. These calculations showed an astonishing agreement with the laboratory experiments, in which a little magnetic model of the earth in a glass vessel, exhausted of air, was exposed to the effects of of cathode rays.

Moreover, he tested his theories in Nature herself, for he undertook no less than three arduous expeditions to the northern regions where the northern lights show especial strength and characteristic appearance. One of these expeditions extended to stations in Iceland, Spitzbergen, Novazembla, and Finmarken, so that by collective observations at the various places the value of the observations might be increased.

In connection with the study of these phenomena Birkeland has been occupied with another great task—the explanation of the earth's magnetism, its variations, daily and yearly, in addition to magnetic storms. Here also Birkeland has shown himself not only a scientific genius, but also an experienced and rigorous experimenter, for without experimental confirmation even the best theory will be weak and vague.

He has, in short, proved his theory with the help of a large number of co-operative observations taken at no less than 25 stations throughout the entire world, and the mathematical utilization of these observations has required an enormous amount of work necessitating the aid of a whole staff of scientific coworkers and the expense of large sums of money. These expenses have for a large part been met by Birkeland himself. Such a sacrifice for science is rare, the more so as he was by no means a wealthy man.

In fact, he procured the money necessary for these purposes by putting aside his purely scientific studies for a period of five years and devoting himself to one of the greatest problems with which chemistry is concerned at the present day, namely, the production of nitrogenous compounds from the atmosphere by the aid of electricity.

After having succeeded with the imagination of genius and with energetic work in solving this problem even as to the details of the first practical method of manufacture ever carried out, and having obtained the money he needed for the abovementioned scientific objects, he again set to work with all his force in the service of science.

Birkeland had noticed during one of his experiments that an electric arc of high potential in a strong magnetic field assumes the form of a flat disk of fire, possessing tremendous heat and power of chemical combination. He spoke about this discovery to an engineer, Mr. Sam Eyde, who had read about the American attempts to produce nitric acid from the atmosphere with the aid of electricity, and the idea arose of using the said disk of fire for the same purpose. Experiments that were then carried out confirmed the feasibility of the scheme, and the systematical development of the process led to the formation of great companies which have factories in Norway for the production of nitric acid from the atmosphere.

The Norwegian Hydroelectric Company (capital 41,166,250 francs, or $8,233,250), which operates under the Birkeland-Eyde patents, started in 1908 the Notodden-Svolgsfos factories with 40,000 hp. This company is also building factories at Lienfos (15,000 hp) and Wamma (55,000 hp). The company has also combined with the "Badische Anilin and Sodafabrik," which has elaborated its own methods in later years. These united companies are now constructing factories at Rjukan (227,000 hp), Tyin (81,000 hp), and Matre (83,000 hp).

Professor Birkeland has been elected a member of the Scientific Society of Christiania, he is a doctor honoris causa of the technical high school in Dresden, and an honorary member of the Royal Society of Arts and other learned bodies.

He is undoubtedly the greatest living scientific investigator of Norway, possibly of Scandinavia. He possesses a deep and broad scientific education, an unusually rich imagination and forceful synthetic power; he is a scientific genius with the happy gift of thinking out and carrying through the experiments that are to prove or disprove his theories. He has, in addition, stored in him unlimited energy, which he entirely devotes to the solution of scientific problems—and he permits no obstacles to deter him from his task.

C. N. RÜBER.

Smoke and Fume Problem in Bayonne.—According to the New York *World*, Attorney-General O'Malley, acting upon orders from Governor Hughes, of New York, is to bring action in the Federal Court against the Standard Oil Company, the Bergenpoint Chemical Company, the General Chemical Company, the Orford Copper Company, and other chemical companies operating in Constable Hook, Bayonne, N. J. Constable Hook is directly opposite to Staten Island, which is a part of New York City, and it is in Staten Island that the damage is alleged to have been done by the smokes and fumes from the chemical and metallurgical works of Bayonne. Since the latter are in the State of New Jersey the legal questions involved are interesting.

The Strength of the Alloys of Electrolytic Iron and Monel Metal.

By Charles F. Burgess and James Aston.

For several years past there has been conducted at the Chemical Engineering Laboratories of the University of Wisconsin, under grant of the Carnegie Institution of Washington, an extensive series of tests on the properties resulting from the addition of various alloying elements to electrolytic iron. As part of the tests, a study was carried out upon the strength of the alloys of this iron with varying proportions of nickel and of copper.

The results of the nickel series were published in Bulletin No. 346 of the University of Wisconsin,[1] and those of the copper series were presented before the American Electrochemical Society in October, 1909,[2] and reprinted in the above-mentioned university bulletin. While this work may in large measure be classed as supplementary or correlative to the work of other investigators,[3] it was considered well worth undertaking in view of the high degree of purity of the electrolytic iron, and the consequent opportunity of ascribing any observed effects solely to the alloying element, and the elimination thereby of one question of doubt arising in most investigations of the alloy steels.

The results of these earlier tests were largely confirmatory of the work of the other investigators mentioned, the chief differences noted in our tests being a shifting of the regions of certain physical properties because of variations of carbon content. The general conclusions may be stated as follows:

Iron-Nickel Alloys.

1. With increase of nickel there is an increase of strength with a slight decrease of ductility in the range of lower nickel content.

2. Beyond this range the addition of nickel causes a sudden increase of strength with a marked decrease of ductility over a zone of decided brittleness.

3. The position of the brittle zone varies with the carbon content and probably with a variation of other impurities, such as manganese, if present in appreciable amounts. For a pure alloy where the effect is due to nickel alone, the area may be set between 10 per cent and 16 per cent of nickel. The addition of carbon places the zone at lower percentages of nickel, as low as 7 per cent nickel for carbon = 0.82 per cent, or even as low as 5 per cent if the manganese rises to 1 per cent.

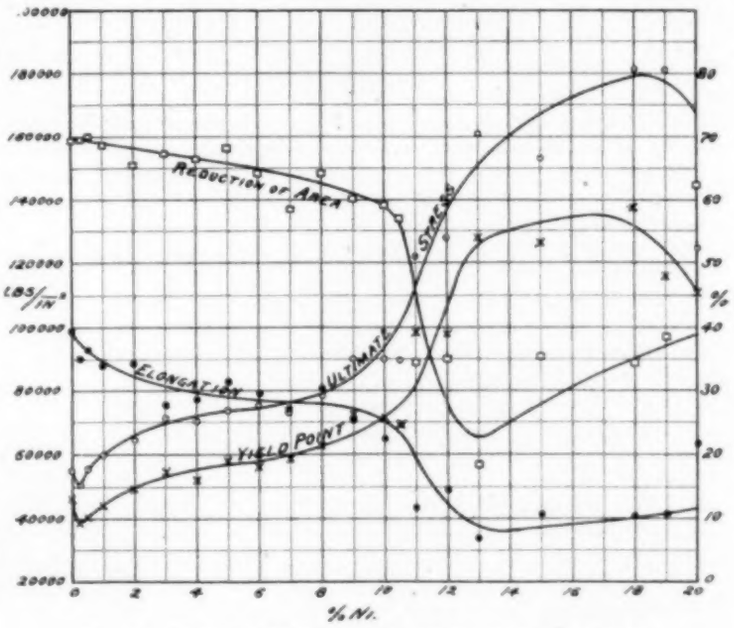


FIG. 2.—TENSILE STRENGTH IRON-NICKEL ALLOYS ANNEALED.

4. Accompanying this brittleness there is a marked hardening in the material.

5. Annealing, while not greatly affecting the region of the brittle zone or the extent of the brittleness, has a tendency to confine the range to more narrow limits.

6. For percentages of nickel above those of the zone of brittleness there is a restoration of the ductility and softness.

The data of this series are given briefly in Tables 2 and 3, and plotted in Figs. 1 and 2.

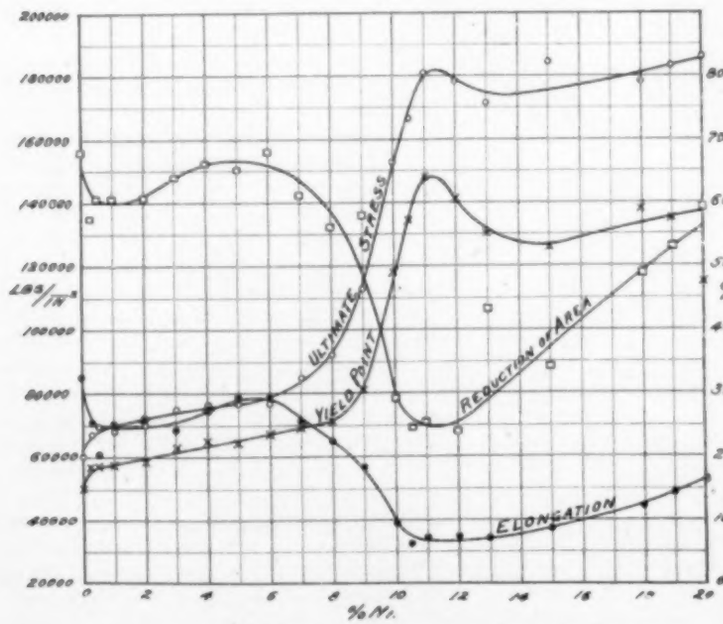


FIG. 1.—TENSILE STRENGTH IRON-NICKEL ALLOYS UNANNEALED.

Iron-Copper Alloys.

The results are tabulated in Tables 2 and 3 and plotted in Figs. 3 and 4. They are in disagreement with the generally accepted opinion that copper is a very deleterious impurity in steels. It seems evident that the detrimental effect commonly noted is due rather to the combined influence of the copper and some of the elements met with in commercial steels, notably carbon and sulphur. In the absence of carbon, unannealed samples showed a successive increase of tensile strength up to the maximum copper addition of 7 per cent (the limit of forgeability). The yield point was close to the ultimate strength, and the elongation fell off with regularity. Brittleness, therefore, was the accompaniment of the copper additions, and increased with the percentage present.

Annealing had a most marked effect, lowering the ultimate strength and elastic limit throughout, but especially for copper percentages above 1. The elongation became very good throughout the entire range. The result was, therefore, an alloy of very good properties, and this fact was brought out in the paper previously referred to. We mentioned particularly the addition of 1½ per cent of copper as being advantageous, because this small amount of addition agent resulted in prop-

[1] The Strength of the Alloys of Nickel and Copper with Electrolytic Iron. C. F. Burgess and James Aston. *Bulletin* Univ. of Wis., Eng. Series, Vol. 6, No. 3, March, 1910. (See Book Reviews in this issue.—Editor.)

[2] Physical Properties of Iron Copper Alloys. C. F. Burgess and J. Aston. *Trans.* Am. Elec. Chem. Soc., Vol. XVI, 1909.

[3] Nickel Steels: R. A. Hadfield, Inst. Civ. Engrs., 1898, Vol. CXXXVIII; R. Guillet, *Bul.* Soc. d'Encour, May, 1903; 7th Report Alloys Research Committee, Inst. Mech. Engrs., Vol. 2, 1905. Copper Steels: P. Breuil, *Jour.* Ir. & St. Inst., No. 2, 1907.

erties almost as good as those obtained with larger quantities of copper, and approached the results reached with the customary nickel content.

Iron-Nickel-Copper Alloys.

The present paper is the direct outcome of the observations noted in the two binary series of alloys just discussed; its inception was due to reasons expressed in the above-mentioned university *Bulletin*, in which some preliminary work was reported and which reasons may be quoted verbatim here.

"In view of the well-known beneficial effect of the addition of nickel to iron, and because of the very great and rather unexpected increase of tensile strength observed, in our tests of the copper-iron alloys, due to increasing percentage of copper, it was thought advisable in order to round out this research to carry out tests on a series of alloys in which both nickel and copper were added to the electrolytic iron.

"The advisability of the investigation was prompted by several considerations—first, the benefits observed by the separate additions of nickel and copper might be coupled in their joint use; second, the close relationship of copper, nickel and iron in chemical and physical properties might lead to interesting results in a ternary alloy; third, (and this was the reason of greatest weight), there is on the market a nickel-copper alloy which would make a very desirable addition agent, in case the simultaneous presence of both copper and nickel gave a tensile strength no less than that observed for the two separate alloys.

"We refer to Monel metal, resulting from the reduction of certain ores from which, by smelting alone ,this alloy is formed. The main constituents are nickel and copper, in the proportions of about three to one, respectively. The material is, for the present at least, obtainable at a price about equal to that of copper, and very much below the cost of nickel. Here, then, is a means of getting nickel additions, provided the copper carried does not neutralize the effect of the nickel."

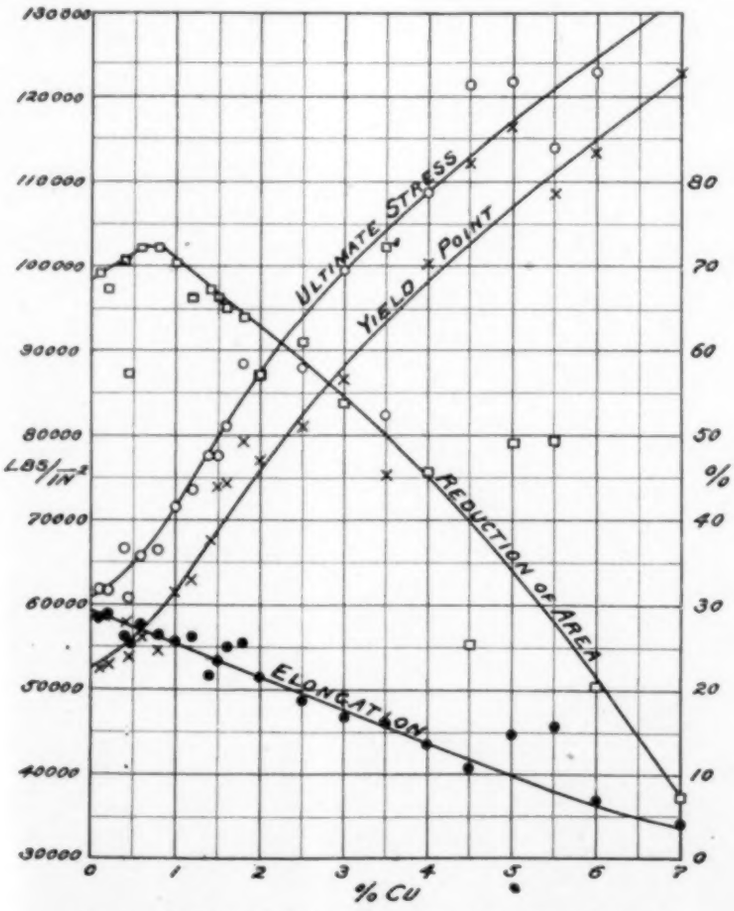


FIG. 3.—TENSILE STRENGTH IRON-COPPER ALLOYS UNANNEALED.

In the preliminary study no Monel metal was available for use. The alloys were made up by additions of nickel and copper up to a total combined content of 8 per cent. The Monel ratio of three nickel to one copper was adhered to, and, strange to say, the proportion of 1½ per cent of copper to 4½ per cent of nickel gives the values approximately the best observed in the separate alloys. The results of this work, given in Tables 2 and 3, and in the dotted curve of Plates 5 and 6, were sufficiently encouraging to warrant carrying the investigation to higher total addition content, and with Monel metal as the alloying agent.

Some particulars regarding Monel metal are given below:

Analyses.

Nickel	66.90	67.96
Copper	24.35	26.00
Iron	5.00	2.80
Manganese	2.18	1.62

Mechanical tests gave for cast samples ultimate strengths of 30,400 lb. and 35,000 lb. per square inch. For rolled samples the figures were: elastic limit, 74,400 lb. and 79,000 lb. per square inch; ultimate strength, 100,000 lb. and 104,000 lb. per square inch.

Thus we see that the alloy has very good inherent mechanical properties, and the analyses indicate, besides the nickel and copper, only iron and manganese, and the latter is no detriment to steels.

The iron used was of a high degree of purity, obtained by a double electrolytic refining of Swedish iron. The electrolytic iron may be considered 99.97 pure. In the preparation of the alloys the brittle iron was broken into small pieces and melted in an electric furnace of the resistor type with the predetermined amounts of Monel metal. Specially prepared magnesia crucibles were used, these in turn being protected by a graphite jacket; and contamination of the charge was prevented by a magnesia lid luted on the crucible and a graphite cover for the jacket.

During the melt there was no agitation of the charge, the resulting alloy being entirely due to diffusion of the elements. This was facilitated by holding the charges above the melting temperature for a few hours, and by slow cooling in the furnace. The ingots weighted about 500 grams (1 lb.), and after forging, each furnished enough material for three test bars approximating 0.4 in. finished diameter over a free length of 2 in. Ingots were made up with the Monel metal increasing by gradations of 2 per cent to a maximum addition of 20 per cent. Samples were tested as forged and after annealing at 900° C.

The tests were made upon a Riehle 50,000-lb. power-driven machine. The load was applied very slowly, and the yield point was determined by means of multiple dividers. This was checked by the drop of the beam.

From 2 to 12 per cent of alloying addition, the ingots forged readily over a wide range of heats; from 12 to 18 per cent high temperatures were necessary and the bars did not forge well below bright red heat, while for those ingots with 18 and 20 per cent of Monel metal a white scintillating heat was necessary. Up to and inclusive of the 12 per cent addition, the test bars were machined in the lathe; above this grinding was resorted to, the material being too hard for efficient working with lathe tools.

Tensile Strength Tests.

The summary of results is included in Table 1 and in Figs.

5 and 6. There is noted the yield point and ultimate strength in pounds per square inch, the percentage of elongation in 2 in., and the percentage of reduction of area at fracture. There are included also the maximum, minimum and average values of each of the above items, together with the number of samples from which such average is calculated. In the plates the yield point, ultimate strength, elongation and reduction of area for varying percentages of alloying additions are designated by appropriate symbols, and are the values given in the tables in the columns of averages.

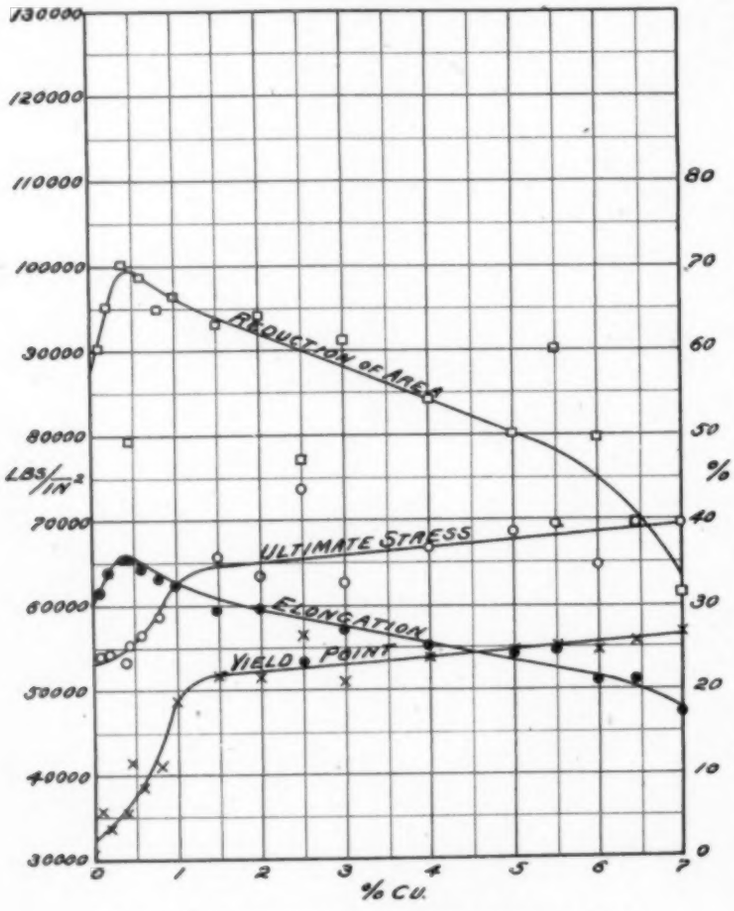


FIG. 4.—TENSILE STRENGTH IRON-COPPER ALLOYS ANNEALED.

With these points as a guide, smooth curves are drawn in each case. With the many factors affecting somewhat the strength of a metal, it is not expected that the points will fall consistently on a curve. It is intended merely that the curve will show the tendency of the effect of increasing the percentage of Monel metal in the alloy.

Unannealed.—The results of the tests in the unannealed state are enumerated in Table 1, and the general summary is best noted by referring to the curves on Fig. 5. Between 0 and 8 per cent there is a rather rapid rise in the ultimate strength and the elastic limit; the latter rises more rapidly, however, and at 8 per cent the elastic ratio is 0.85, indicating brittleness. This is confirmed by the falling off in the values of elongation and reduction of area. Up to 4 per cent of Monel addition there is an increase of ductility, and at this point we note the very fair values of ultimate stress 81,000, yield point 61,070, elongation 30 per cent, and reduction of area 64.3 per cent.

From 8 per cent upward there is a rapid increase in the maximum strength and elastic limit, but with ever-increasing brittleness, as is indicated by the steady falling off of the curves for elongation and reduction of area. The highest values reached for the series were for one of the bars with 18 per cent of alloying addition, the figures being: ultimate, 215,000; yield point, 172,000; elongation, 10 per cent, and reduction of area, 27.6 per cent.

Annealed.—These data are noted in Table 1 and Fig. 6. In the alloys of lower Monel additions annealing has had but little effect on the properties. In fact, and this was noted in the preliminary iron-nickel-copper series, the tendency has been to increase the ultimate strength and elastic limit without material falling off in the ductility. For those alloys with 4, 6 and 8 per cent of addition (the probable useful range) the strength is higher. We now note an increase of ultimate stress and yield point in almost a linear function of the amount of alloying material until at 10 per cent the values are: ultimate, 150,000; yield point, 144,400; elongation, 15.7 per cent; reduction of area, 51.9 per cent. The elongation and reduction of area are in accordance with the stress values.

Between 10 and 20 per cent of Monel metal the ultimate strength undergoes but little variation, while the elastic limit becomes less with increasing additions. Strangely enough, the elongation and reduction becomes steadily less, the latter in particular falling off very abruptly at the higher percentages.

Comparisons.

Direct comparisons of the general tendencies in the iron-Monel metal alloys is best made by means of the curves of Plates 1, 2, 5 and 6. In the unannealed condition (Figs. 1 and 5) there is a marked similarity, considering equal percentages of total additions. In each case there is a maximum of

TENSILE STRENGTH TABLE I.
ELECTROLYTIC IRON-MONEL METAL ALLOYS.

Per cent Monel.	No. Sample	Yield Point lbs. / sq. in Max.	Min.	Av.	Max. Stress lbs. / sq. in Max.	Min.	Av.	Elongation % / 2" Max.	Min.	Av.	Reduction Area % Max.	Min.	Av.
UNANNEALED													
2	1			49300			71000			22.5			62.5
4	3	67400	57000	61070	88700	77000	81000	30.0	30.0	30.0	69.0	61.5	64.3
6	1			72500			84000			25.0			54.0
8	3	92500	91400	91830	118300	117200	107670	20.0	20.0	20.0	55.2	51.8	53.2
10	3	150500	132000	139830	170000	158000	163000	15.0	10.0	12.8	42.6	28.6	37.1
12	3	161000	153500	157500	189000	180500	185400	13.5	12.5	13.0	44.5	36.1	40.4
14	4	152000	130000	144370	182000	157300	166820	10.0	5.0	6.9	30.0	8.5	16.4
16	3	171000	154000	162500	191500	179000	184330	12.5	5.0	9.8	37.3	12.2	27.2
18	2	172000	168200	170100	215000	196000	205500	10.0	5.0	7.5	27.6	7.2	17.4
20	2	191000	183200	187100	205000	202000	203500	7.5	4.5	6.0	8.8	5.8	7.3
ANNEALED 900°C.													
2	2	40500	40000	40250	64500	61500	63000	30.0	30.0	30.0	70.0	58.0	64.0
4	2	71600	71000	71300	91000	84600	87800	25.0	22.5	23.7	61.0	59.5	60.2
6	2	94000	88000	91000	109000	105600	107300	20.0	17.5	18.7	51.7	47.3	49.5
8	3	98500	75800	88100	115200	90700	102970	21.0	18.5	19.8	64.6	59.0	62.5
10	3	158000	126200	144400	165000	131000	150000	17.0	15.0	15.7	59.8	45.0	51.9
12	3	160800	120000	145930	167200	126000	151400	15.5	10.0	12.2	50.5	31.6	39.0
14	2	150000	143600	146800	156000	147000	151500	10.0	10.0	10.0	40.7	36.0	38.3
16	3	152000	120000	138000	166500	142000	153000	11.5	10.0	10.8	41.6	39.5	40.5
18	2	157000	103000	130000	166500	158000	162250	13.5	7.5	10.5	29.4	11.9	15.7
20	2	130000	123400	126700	170000	139900	154950	5.0	2.5	3.8	1.1	1.0	1.1

strength and elasticity at about 12 per cent. Up to this quantity there is a continual rise, more abrupt, however, in the case of the Monel alloys; and beyond 12 per cent there is the same dip in the curves with subsequent rise. On the other hand the nickel alloys show a brittle zone at about 12 per cent, with a recovery of ductility with further additions; with the Monel metal alloys the brittleness becomes more pronounced with increasing additions.

The relations in the annealed samples (Figs. 2 and 6) are similar to those commented upon above. The same brittle zone with subsequent recovery is noted in the nickel series, but in the Monel metal alloys the brittleness increases continuously. Up to 10 per cent of total additions the increase of ultimate strength and elastic limit is much more pronounced with Monel metal than with nickel alone, but the ductility is greater in the latter case. In both cases the pronounced maximum of strength and elasticity is now not noticed. On the other hand, while annealing of the nickel alloys has lowered the ultimate strength and the elastic limit throughout the entire range, with an accompanying increase of elongation and reduction, the effect of heating of the Monel metal series has been pronounced only for percentages above 10, below this range the result is to increase the strength somewhat with little alteration of ductility.

TABLE 2

COMPARISON IRON-NICKEL-COPPER ALLOYS

UNANNEALED

Per cent Addition	Yield Point				Max. Stress				Per cent Elong.				Per cent Red.			
	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni-Cu	Cu
1.5				73920				77300				23.5				66.9
2.0	58520	49300		76930	70350	71000		87010	26.0	22.5		21.2	60.7	62.5		57.0
2.7			62100				79200				26.0				56.7	
3.0	62920			86810	74860			99640	24.5			16.7	63.9			53.8
4.0	65000	61070	67400	100560	75670	81000	80100	108640	27.8	30.0	28.0	13.5	66.5	64.3	58.8	45.6
5.0	64030			116350	77150			121900	29.2			14.5	65.3			49.2
5.3			63300				85700				23.0				45.7	
6.0	67300	72500		113400	77300	84000		122900	29.0	25.0		6.5	68.5	54.0		20.3
6.6			66500				89400				22.0				39.3	
8.0	70770	91830	81700	135100	91740	107670	99700	148750	22.6	20.0	25.0	0	56.1	53.2	53.5	0
10.0	118960	139830			153360	163000			9.1	12.8			29.0	37.1		
12.0	141140	157500			178420	185400			7.1	13.0			23.8	40.4		
14.0		144370				166820				6.9				16.4		
15.0	125800				184730				8.2				34.1			
16.0		162500				184330				9.8				27.2		
18.0	138170	170100			178500	205500			12.3	7.5			49.3	17.4		
19.0	134730				183270				14.2				53.1			
20.0	115500	187100			186100	203500			16.0	6.0			59.5	7.3		

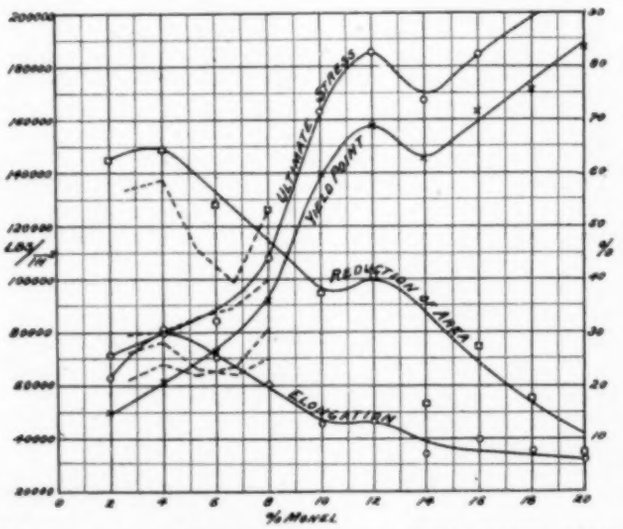


FIG. 5.—TENSILE STRENGTH IRON-MONEL METAL ALLOYS, UNANNEALED.

In Tables 2 and 3 are collected the data of all of the series in order to make direct comparisons of the effects of the addition of Monel metal and of nickel and copper to iron in the ternary alloys, with those of the separate additions of the two elements in the binary series. The several items are noted in columns for each group; the compositions are given as the total percentage of alloying addition; the separate proportions of nickel and copper in the ternary series can be calculated from an assumed ratio of three nickel to one copper.

In the preliminary work the object was to see whether the properties ordinarily accompanying the addition of nickel to iron were detrimentally affected by the copper carried in Monel metal and the comparison was made on the basis of equal nickel content and regarding the copper as so much superfluous addition. However, it was found that the copper had a beneficial effect, and a more rational comparison, therefore, can be made by tabulating on the basis of total addition material.

But little discussion of the figures is necessary. It will be noted that the ternary alloys (both Monel and nickel-copper) hold their own with either of the binary alloys through the entire range. And this is especially true in the annealed samples, particularly below 10 per cent of total additions, the range of probable greatest utility. At 4 per cent, for example, we note the following figures: ultimate strength, 87,800 and 70,070 yield point, 71,300 and 52,100; elongation, 23.7 and 28.4, and reduction 60.2 and 66.7, these figures being, respectively, for total Monel metal and nickel additions. The ductility of the

TABLE 3

COMPARISON IRON-NICKEL-COPPER ALLOYS

ANNEALED

Per cent Addition	Yield Point				Max. Stress				Per cent Elong.				Per cent Red.			
	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni Cu	Cu	Ni	Monel	Ni-Cu	Cu
1.5				51750				65720				29.2				63.1
2.0	48770	40250		51620	64400	63000		63200	34.2	30.0		29.5	65.4	64.0		64.4
2.7			48600				68700				30.0				61.5	
3.0	55400			50890	70670			62360	27.5			26.9	67.6			61.7
4.0	52100	71700	56800	53570	70070	87800	72500	66540	28.4	23.7	31.0	24.8	66.7	60.2	59.1	54.2
5.0	58320			54460	73230			68250	31.4			24.2	68.1			50.3
5.3			67600				84700				28.0				55.6	
6.0	56400	91000		54420	75300	107300		64530	29.5	18.7		20.8	64.2	49.5		49.9
6.6			74200				86400									
8.0	63320	88100	83200		78470	102970	103700		30.2	19.8	22.0		64.4	62.5	56.2	
10.0	71800	144400			89300	150000			22.3	15.7			59.1	51.9		
12.0	97730	145930			121800	151400			14.5	12.2			35.8	39.0		
14.0		146800				151500				10.0				38.3		
15.0	126300				153300				10.5				35.8			
16.0		138000				153000				10.8				40.5		
18.0	137700	130000			181550	162250			10.0	10.5			34.4	15.7		
19.0	116150				180700				10.7				38.7			
20.0	111000	126700			124900	154950			21.5	3.8			62.5	1.1		

Monel samples is entirely comparable to that of the nickel series in the unannealed condition; after annealing it is somewhat lower. But this may be due to the necessity of different annealing temperatures since in the ternary nickel-copper-iron series higher values for elongation are the rule.

These results seem to warrant the conclusion that for the carbon-free alloys, at least, the combined addition of nickel and copper in the proportions found in the commercial Monel metal does not result in any deterioration of the good qualities

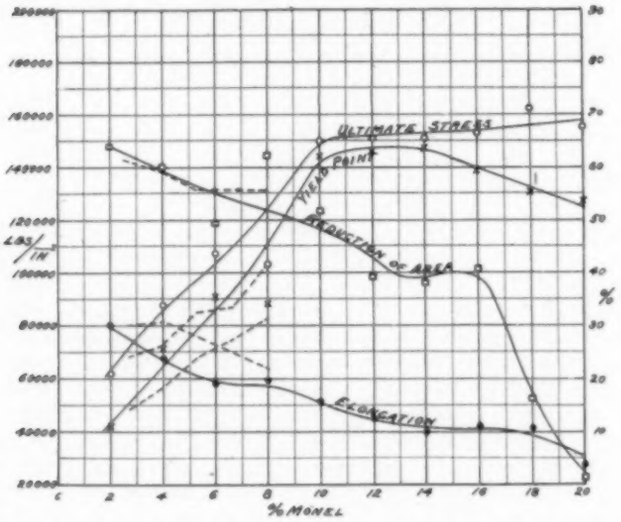


FIG. 6.—TENSILE STRENGTH IRON-MONEL METAL ALLOYS, ANNEALED.

conferred by the nickel alone; in fact, the copper has good inherent qualities of its own. The general effect is the same for both series for equal total alloying additions. Whether these same good features will result from like additions to commercial steels, with their customary amounts of impurities, is a problem now under consideration.

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Corrosion of Iron and Steel and Its Prevention.—A paper read by Mr. Gustave W. Thompson, of Brooklyn, N. Y., before the American Institute of Chemical Engineers gives a summary of the facts which have been discovered with reference to the cause and prevention of the corrosion of iron and steel. The author leaves it to the reader to interpret these facts in the terms of any theory with which he may be familiar. The general conditions conducive to the formation of rust are first given. The opinion is given that facts do not warrant the conclusion that the tendency of iron to corrode is dependent on the composition of the metal and that the use of a poorer and more expensive metal as against a more impure and cheaper metal is not justified by the facts. Practical suggestions with reference to the prevention of corrosion of iron and steel are given, among which the following may be mentioned: The action of acids, moisture and oxygen. Cement prevents corrosion because it is alkaline. Protective coatings exclude moisture and oxygen. The cleaning of iron and steel before painting is of great importance. Specific rules for the cleaning of steel and iron are given.

The principles which should guide in the selection or designing of the paint are given. Poor workmanship in the application of paint was given as responsible for most of the failures in the protection of iron and steel with paint. Linseed oil was condemned as a prime coat for iron and steel. Nineteen different paints applied to the Havre de Grace bridge of the Pennsylvania Railroad all proved equally effective because they were properly applied. Oxygen cannot be excluded from iron by means of any protective coating. In the absence of moisture, oxygen does not produce corrosion.

Study of Materials as a Subject in a Course in Chemical Engineering.

Dr. Charles F. McKenna's presidential address delivered at the Niagara Falls meeting of the American Institute of Chemical Engineers called attention to the rapid development of the studies whose aim it is to increase our knowledge of the "properties of materials." In the past, there has been a limited view of both the particular objects to be studied under the term "materials" and also of the scope of the inquest to be constituted into the properties. Elasticians began their studies upon beams and the problem of Galilee led to consideration of compression of fibers and thus to the study of columns and other forms used in construction. Naturally, the properties studied were the strength elements and the materials considered only those used by constructors.

But a broader view of both has now been brought about by two influential factors. First, the deep researches of mathematicians into the theories of elasticity have extended that study until it is seen to be founded on conceptions of cohesion molecular constitution, intermolecular attractions or other forces and that these lead far afield amongst many other new territories of physics. Secondly, modern activities have developed engineers who busy themselves in many other branches than construction or the art of war, which once monopolized their science, and who utilize as substantial agencies many other materials than wood or iron or lead. This is a return to a better practice which was in vogue in more ancient times, as is shown in some remarkable passages in literature, as in Pliny and Nitruvius, not to go back to the Book of Books.

An extraordinary number of substances are now-a-days being made the subject of inquiry by the engineer as well as by the technologist. But there are great dangers in the pathway of those who essay to study those materials in the haphazard fashion of an untrained observer, or to use them without any proper study of their nature.

As generally understood "materials" are those substances or fabrics whose applications are found in the useful arts. Raw materials form the basis of manufacture and the results are besides the chief product desired, by-products and waste materials. It is the aim of technology to so perfect materials and processes that there should be no wastes and that all products be brought into such conditions or such markets that revenues can be obtained from each adequate to its importance. In the history of some industries, it is often noted that the waste product of former days is now almost as much sought after as the former chief product, indeed, in some instances has preempted that position.

"Materiology" can be defined as the study of the properties of materials applicable in the useful arts and of the method of testing and examining them for the determination or comparison of these properties.

Properties may be real or virtual. Real properties are those which belong, in the first instance, to the substance itself, and by which we recognize it in both matter and form. Virtual properties are those which a material has by virtue of its reaction with another material. Composition, density, specific heat are real properties of gasoline, but the property of exploding in a closed cylinder is only a virtual one, for it cannot be manifested without the presence of air. Lubricating oil possesses in viscosity a real property, but its lubricating value is a virtual one, because it is effected by many other conditions.

That this distinction makes for clearness and is not a mere refinement, is illustrated by the case of coal dust and flour dust, which some might class together as explosives. But true explosives are those which contain within themselves the oxygen necessary for complete combustion, while organic bodies, such as coal dust or flour dust require air in intimate mixture before they can become explosive; it is, therefore, the mixture which is really explosive, the particular dust being only virtually ex-

plosive. Whether real or virtual, we recognize chemical properties, physico-chemical ones, physical ones, mechanical resistances, and structure.

In chemical properties we have to consider the active and the passive properties, the ultimate constitution or the proximate constitution. In some industries, as in the corn industry, the ultimate chemical constitution of the chief materials is of fundamental importance; in others, as in fertilizers, rubber gum, etc., only the proximate constitution is of importance.

Physico-chemical properties are seldom studied outside of the research laboratories, but classified knowledge of this character is of primary importance for the chemical engineer.

Under structure the materials are treated of microscopically and macroscopically, with regard to their morphology and state of aggregation. Few achievements of modern times in applied science have surpassed in importance the work done by this mode of study in alloys.

To acquire a full knowledge of the most intimate properties, direct and indirect, real and virtual, of all technical substances, we must approach the subject with a peculiar mental equipment, which can only be obtained by much experience. But there are some general principles which supply the necessary universality yet make for simplicity and order.

(1) The full and true value of a material for use in the arts can only be determined by ascertaining the nature of the substance or substances composing it.

Very often a material is applied to certain purposes after a very superficial study and a realization of the inappropriateness of the choice comes too late. The crumbling walls of limestone or of brownstone seen in New York City are examples of this. Other examples are found in the use of refractory clay materials. A fireclay can on certain rough tests be classed as refractory; a real appraisal of its value can only be made by using refined method of chemical analysis, mineralogical or rational analyses, extended fire tests, shrinkage tests, drying tests, etc., observations of conditions, structure, etc.

(2) The simple or homogeneous substances must be the basis of study before the nature of the complex or heterogeneous can be understood.

We should know the character of iron before we presume to understand the character of iron and carbon, iron and manganese, iron and nickel, etc.

Portland cement furnishes us with another example of the applicability of this principle. In this case we should first study lime and its hydrate, then the properties of the silicates and aluminates and sulphates, their interractions, and even, as well as those of the minerals or substances in nature having the same or analagous compositions, besides bringing to bear upon it a full comprehension of the conditions surrounding matter in the colloidal state.

(3) The economic adaptability of a material can only be determined by a knowledge of its behavior under all the forces inherent or applicable to it.

Tin would be an example of this, which we look upon as a firm and moderately resistant metal at all ordinary temperatures. It would be well to know, however, that if it is to be used in some very cold position it can revert to the enantiotropic form in which it has no cohesion and falls to powder. This has lately been made the basis of a detinning process.

(4) Where a series of causes operate upon a material, their effects cannot be understood until these causes are recognized and investigated, and their operation in order of time understood. Dr. McKenna made this clear by giving the case of fireproof wood as an example. Not many years ago an investigation of fireproof wood was projected in the hope of finding a satisfactory and discriminating test, and the committee assuming charge of the matter proposed to study the different treating processes and deduce some views of testing the relative fireproofing value as well as the incidental effects of poisonous gases driven off by heat from the chemicals saturating the wood. Various ways of applying the flames to shavings and to test sticks were to be assayed. Strength tests of the wood before and after treatment were to be made and various such features were to be investigated. This investigation was never consummated, but it is probable that if it had been made according to this program much futile effort would have been spent upon it.

It was early pointed out that the proper study depended upon a previous knowledge of what really goes on when wood is subjected to heat and flames. This heat can be assumed to be initially low and progressively rising. The conditions in some cases, particularly regarding the exterior surfaces, would be those normal to a free kindling of wood. The conditions as regards other portions would be those prevailing in the destructive distillation of wood without access of air. Naturally, therefore, a long series of heating and distillation tests would be necessary, and few of the other tests of the previous program would be of any value. As a matter of fact, such a study, using a small electrically heated retort adapted for quick work, did show that the yield of gas from the heating of wood in closed vessels would make an excellent indicator of the efficacy of the fire-retarding salts used for treatment, and would not give the contradictory results of the shavings or fire test. Many complex processes in technology would become research problems of the first order, and with great promise of positive additions to knowledge, if this principle were observed.

(5) Purity of substances and behavior of materials under the action of forces can be formulated in terms called Standard Materials, but cannot be determined in individual specimens with greater exactness than the limitations of observations, of instrumental precision and of the accuracy of the available standards of measure will admit.

While this simply means that errors are personal and instrumental, it is well at times to keep it in full view and to try to know well the equations and their quantitative relations. One feature of this principle was covered by Ostwald in what he called "the fundamental rule for drawing up a plan of measurement: The accuracy of the measurement must correspond to the exactness of the definition of the object to be measured."

A recognition of this principle also will save much useless work, because it is, unfortunately, too common to see enormous numbers of experiments proposed and numbers of workers engaged therein striving to use much refinement of measurement while the definition of the point involved is on a scale so much greater that no measurements will be needed other than what could be called gross measurements.

This principle also contains the germ of an idea which has often occurred to Dr. McKenna in discussing the vexed question of standard or uniform methods of analysis. Too great an insistance upon the adoption of these methods is apt to lead away from truth rather than toward it, and apt to put a premium upon too little investigation rather than to show the danger of too great effort or waste of too much time. Every substance of the same name has not the same composition, and any method which turns the chemist into a follower of system may produce economy of time in the routine of a factory and may more often than not favor comparison of results by diverse interests; but it is conceivable that the truth can thus be missed at times, and it is quite certain that in ultimate reference more independence must be permitted to the experienced analyst.

(6) In obtaining data on mechanical resistances the mode of application of a stress must be adequate, and the means of measurement of the strain adapted to the derivation of a factor commensurate to the magnitudes to which that factor is to be applied. In a great many branches of experimental science we can assert that what is true of the small is true of the large. This is more often observed in chemistry than in physics; it is most often physical factors which introduce variables into a problem when the results of a small operation are to be applied on a larger scale.

In pyrometry and the study of fuels we find that calorimeter test results of a coal do not coincide with results obtained on the large scale with the steam coal in a boiler test.

But in mechanics it is particularly necessary to consider this element of variation. For example, when calculating upon safety factors in structures whose parts are so gigantic as to make difficult the use of coefficients by the mere multiplication of cross-sectional areas. The Quebec Bridge was designed and almost completely erected upon the assumption of the correctness of certain factors for the resistance to compression obtained in laboratory tests. No data was available for computing from results obtained on large members. This was a fatal element of weakness and led to the disastrous collapse of the structure.

In the **"control of materials"** certain distinctions should be observed and certain authoritative positions should be assigned to the individuals concerned in these important affairs. Thus, each of the following functions are distinct: Specification, inspection, testing, judgment, sanction.

The drafting of a specification for material is the work of an engineer, an architect or technologist, making use of his experience and knowledge, or the knowledge of an authority on materials. The inspection is the routine of determining the compliance with the specifications of the material being delivered; testing preliminary samples submitted from outside sources is not inspection. Inspection involves the sampling and weighing of cargoes, determination of observable and important conditions at the time of delivery, and the transmission of samples intact to the testing laboratory. Inspection is not testing, nor is testing itself complete till the judgment is rendered as to values. Finally, this preliminary work would be useless if there could be no enforcement to the provisions of the specification. This is a *sanction* which must go by the act of acceptance, or rejection, and must be within the legal definition of the engineer's powers, as well as be susceptible of support upon referee tests.

The legal aspects bear upon the responsibility of the individuals engaged in this chain of duties. The engineer specifies; the inspector reports deliveries and samples them; the testing engineer makes return under chemical, physical, microscopical or mechanical tests, and states the degree of compliance with the specification; the engineers accepts or rejects. If a contest ensues and it is concerned with the question of accuracy of testing methods, a referee would be chosen from another laboratory, or, if the dispute was to be carried higher, we would look upon the Bureau of Standards as the final arbiter.

In conclusion, Dr. McKenna pointed out that the valuable element in this study for the young chemical engineer is the development of logical processes of thought applied in these practical fields. The great number of hours which would seem to be required for laboratory work would not be a disadvantage when it is realized that this can be co-ordinated in any good course in chemical and physical experimentation and the measurements of science. The student would know the chemical properties of substances in general, would be familiar with the fundamental concepts of the states of matter, but in this study he would consider substances, materials and fabrics in a scheme embracing co-ordinate sciences and in which the mode of attack upon the subject is always one relying upon definitions and distinctions, first principles, exactness of observation and cautious summary and judgment.

If he develops to be a skilled experimenter, manipulator and mechanician he will be especially well qualified. His powers of adapting his resources to his needs for determining the factors of materialogy in any problem before him will give him the same generous equipment of mental self-reliance that a good course in chemical analysis has given him in chemical manufacture. Any study which brings new knowledge into orderly array and reconciliation with historic knowledge, which makes insistent demands upon logical precedure, accuracy of observation and honesty of statement, and then applies results of investigation in a practical way to the most useful works of construction and production is a fit study to place high in a course of chemical engineering.

The Development of Heavy Gravitation Stamps.*

By W. A. Caldecott.

The history of ore crushing by means of gravitation stamps shows a progressive increase in their weight and in corresponding efficiency. This holds good in the development from the square, wooden non-rotating stamps in use in Germany in the Middle Ages to iron-shod wooden stamps in Cornwall, and then to rotary iron stamps in California and mild steel stamps in South Africa. The latter have greatly improved during the last few years, and probably represent the highest existing stage of development. About 1835 the first stamp mill in the United States was started in Tellurium, Virginia. The stamps weighed 50 pounds and were made of square wooden stems with iron shoes and dies, the cams operating in slots in the stems.

Two years later the Vaucluse mine installed 380-lb. stamps, of which the head weighed 125 lb. When the first batteries were erected on the Rand, over 20 years ago, Californian practice probably represented the highest state of the art, and was introduced upon these fields by J. S. Curtis, J. H. Hammond and other well-known Californian mining engineers.

To illustrate the type of battery in operation on the Rand in 1899, in which year the writer's experience in stamp milling began, the following details may be given of the Du Preez Gold Mining Company's mill, premising that this was before the introduction of the cyanide process, and that concentrators, other than blanket strakes and buddles, were not in common use.

The mill was erected in that year and was of Sandycroft make, designed in accordance with the best Californian practice to date. It consisted of 20 stamps weighing 900 lb. each when new, and was run at the rate of 90 7-in. drops per minute. The stamp duty was about 3 tons per 24 hours, using a screen of 900 meshes per square inch.

By 1899 some progress had been made, and 1250-lb. stamps were in use at the Robinson Gold Mining Company and elsewhere, though in Australia and the United States 900 lb. per stamp remained the almost universal maximum limit, and this remains true of the celebrated Homestake mills at the present time. At the Mount Morgan Gold Mining Company (Westralia), in December, 1902, 30 stamps weighing 1500 lb. each were erected. These were eclipsed, however, by the neighboring Millionaire Gold Mining Company's battery of five stamps weighing 1750 lb. each. The advance made on these small companies was not generally realized, and in 1907 few stamps dropping on the Witwatersrand weighed above 1250 lb., with the exception of certain batteries of the Consolidated Goldfields Companies.

In 1904 the author was authorized to carry out a series of milling tests at the Knights Deep battery, which was equipped with stamps weighing 1350 lb. when new. The object of the trials was to discover some means whereby stamp-milling efficiency could be increased.

The initial experiments were made to test the effect of fine breaking before stamp milling, and also whether the increased discharge area afforded by a double discharge mortar box increased the stamp duty. A pair of rolls was installed to effect the preliminary fine breaking, and also a double discharge mortar box. The tests extended over more than a year, and many variations of conditions were investigated, such as varied angles of screen frames, heights of discharge and of drop, and different methods of water introduction into the upper portion of the mortar box.

The results of the trials were almost identical within the

*Abstract of a paper read before the Inst. of Min. & Met. and published in the *Bulletin* of Aug. 12, 1909; also of discussion before the Chem. Met. & Min. Soc. of S. Africa, published in the *Journal* of April, 1910.

limits of experimental error. The average stamp duty on ordinary mill feed for six sets of trials was 5.82 tons with a double discharge mortar, and the same with a single discharge mortar. The average stamp duty for four sets of trials on the product from rolls set ½ in. apart was 5.64 tons with a double discharge mortar and 5.78 tons with single discharge mortar. There was no increased duty to compensate for the extra cost of preliminary fine breaking, and the only difference with the double discharge mortar was a considerable increase in the amount of water used.

The last conclusion is in accordance with the fact that the ordinary single-battery screen has about 40 per cent of its area available for discharge or say 1½ sq. ft. of discharge area for the small stream of pulp yielded by five stamps.

Finally it became obvious that to crush more rock it was necessary to strike a heavier blow. This was sought by fixing helical springs round the stem above the tappet so as to be compressed by the upper guide during the stamp's rise and so that when released the stamp would fall more rapidly, rendering a higher drop and harder stroke practicable. But the life of the springs did not promise to be long and it was finally concluded that a heavier stamp was the best course to adopt.

A series of runs was then made in which direct comparisons were made of the actual tonnage milled by alteration in the weight of the stamps. An increased tonnage was obtained by increased weight of stamp on ordinary mill feed, both under ordinary milling conditions and with a high discharge and back-water feed to obtain a very fine product.

Another run with 1337-lb. stamps, with a height of discharge of 2 in. and a screen of 64 meshes, gave a duty of 10.23 tons per 24 hours, of which 52.8 per cent remained on a 60-mesh screen.

Encouraged by the results of these and many other tests, including a duty of 6.5 tons through 1500-mesh screening with stamps weighing up to 1622 lb., the writer recommended that the next 360 stamps of the Consolidated Goldfields Companies should have a weight of 1550 lb. each when new. Such a departure excited much adverse criticism among experienced millmen generally, on the ground that no foundations could stand the shocks, but the reasoning that if foundations could be built to withstand the blow of the colossal steam stamps, it would be possible to build them to endure the blow of gravitation stamps proved to be correct. All of these heavy stamps are working satisfactorily to-day after a service ranging up to three years and the foundations promise to last as long as did those of their lighter predecessors.

In accordance with the present changed state of professional opinion, all Rand batteries lately erected or designed have stamps of a weight not long ago deemed extravagant, and, aided by the installation of tube mills, stamp duties of 8 tons are as common as of 4 tons a few years ago. While in the United States and Mexico the relatively light stamp is still generally upheld, there is at least one notable exception in the recently built mill of the Boston Consolidated Copper Company,[1] which is equipped with 312 single Nissen stamps of 1500 lb. weight each and a duty of 9 tons.

Somewhat later than the battery tests just given, the subject of battery foundations was investigated. A series of 23 comparative trials was carried out on two five-stamp batteries side by side in the Simmer East mill. One battery was on a concrete foundation only, while the mortar box of the other rested on a cast-iron anvil block placed on a concrete foundation. The results of the trials were practically identical. The tests included screening from 100-mesh to 140-mesh, and the cast-iron anvil block battery showed a duty of 6.84 tons per stamp as against 6.78 tons per stamp on the concrete blocks. One mortar box at the Simmer East mill was erected on a concrete block with a thin layer of insertion between, and has been in commission for three years with no visible sign of deterioration or cracking.

[1] As stated in our issue for June, 1910, p. 313, these stamps are to be discarded as a means of crushing, and will be superseded by rolls and Chilian mills.

Impact vs. Abrasion.

Crushing is usually performed either by impact or abrasion, and while hard ores are dealt with by the former method, the latter is applicable to softer material. The relative efficiency of the two methods for hard material is well illustrated by the diamond, which is hardly capable of abrasion at all except by its own dust, whereas if placed on an anvil it may be shattered into a thousand pieces by the blow of a hammer.

The crushing of larger pieces of ore in stamp milling is done as the result of reaction from the resilient die, the waves of compression transmitted through the ore from the shoe being reflected from the die until the strains set up from the oncoming and reflected waves exceed the elastic limit of the or and rupture results. The stamp is essentially an impact machine, and hence readily pulverizes hard, tough ore. Such ore if handled in any of the legion of rubbing and shearing appliances would be reduced only at the expense of serious wear by abrasion of costly steel parts, which may have to be discarded when only a fraction of their weight has been worn away. With soft material the case is different, and machines of the latter class may be employed with advantage and without undue abrasion of wearing faces.

It is much to be regretted that no satisfactory machine for testing the capacity of material to resist abrasion, or conversely of testing the abrasive capacity of materials requiring reduction, yet exists, as this property is of prime importance in all crushing operations.

When gravity stamps are used for very fine crushing they pass beyond the economic range, and operate by abrasion rather than by impact, with consequent reduction of efficiency as may be seen from the following considerations: Impact is essentially dependent on the pressure exerted by the blow, and depends on the weight of stamp and height of drop. Assume that a 1750-lb. stamp having a 9.5-in. diameter shoe, with a set height of drop of 8 in., strikes a layer of ore 2 in. thick on the die and comes to rest 0.5 in. from the die. Then, having fallen 7.5 in. and having done work for 1.5 in., it has exerted on the ore a mean total pressure of 8750 lb. The shoe having an area of 71 sq. in., the average unit pressure exerted over the whole of its area is only 123 lb. per square inch, or less than that of an average man supported on 1 sq. in. The crushing effect of such a pressure on hard quartz grains is trivial, and a certain amount of reciprocal abrasion of grains of sand rearranging themselves under pressure is probably the main effect produced. The case, however, is quite different in the first impact of the stamp on the ore in the box. Assume that only a 2-in. cube resting on the die is first struck; having an area of 4 sq. in., the average unit pressure here is 2190 lb. per square inch, or 18 times as much as under former conditions.

The finer the crushing is carried in the battery the longer sand particles remain before being reduced sufficiently to escape, and the higher the ratio of fine to coarse material in the mortar box. Hence, under these conditions the liability is for the falling stamp to have its force of impact distributed over so large an area as to produce little effect but abrasion on a bed of shifting, compressible sand. These conditions serve to explain also why fine breaking before milling with heavy stamps does little good. A large number of pieces of ore of approximately equal size afford a large area to receive the blow of relatively small pressure on each. And further, the abrasive action above referred to explains why stamp milling with a fine screen and a high discharge, or both combined, is inefficient through the waste of energy in converting fine sand into slime by abrasion of the particles rearranging themselves under the stamp.

Size of Ore for Stamp Mills.

As regards the desirable size to which ore should be broken

before entering the mortar box, the disadvantage of too fine crushing yielding a uniform bed has already been pointed out. The maximum limit varies with the class of ore, being less for hard ores, but, in general, it should be such that not more than one blow of a stamp is required to pulverize the largest piece of rock. Hence, the heavier the stamp, the coarser the preliminary breaking admissible, and vice versa. Larger pieces are more economically reduced by the rock breaker, and the lowering of stamp duty, owing to less actual height of drop is thus avoided.

Since greater attention has been directed to the use of heavy stamps, it has been customary to use a compensating weight on light stamps. Probably the earliest form of this device was placing an old head at the top of the stem or an extra tappet above the one in use. These devices are crude and do not add the weight where the writer believes it is better, viz., near the bottom of the stamp. The most convenient form of compnsating weight is a split cast-iron disk about 4 in. high, weighing about 50 lb., which can be clamped on the stem by bolts. As many can be added as desired.

With increased weight of stamp, the question of increased shoe and die area arises, but the considerations already advanced, showing the small amount of pressure per square inch of total shoe area, indicate that very large shoe and die areas are not needed for hard ores, while for soft ore a lower and quicker drop can be used. An increase is desirable when the total force of impact of the falling stamp requires a thick layer of ore on the die to avoid excessive shock. Such heavy feeding lessens the duty because the thicker ore bed reduces the height of drop, and the blow is likewise cushioned so that the stamp crushes by abrasion rather than by impact. The cushioning by fine particles will increase with the height of discharge and the fineness of the screen.

The greater the force of impact per unit area, the coarser the screen required for maximum efficiency, as otherwise a thick layer of fine ore particles would be required to avoid pounding. The least thickness of ore layer necessary to take the impact of the stamp without pounding is sufficient to supply ample material for pulverization.

Consequent upon the introduction of the tube mill, coarse battery screening has generally replaced the finer mesh. The tube mill has, moreover, lessened the importance of the battery screen as the criterion of the degree of the reduction of an ore, and the grading analysis of the pulp as it leaves the overflow of the hydraulic classifier is now the guide to the metallurgist as to the degree of reduction. This change makes the degree to which a particle is reduced dependent upon both its diameter and its specific gravity, instead of on the former only, with the result that the pyritic material undergoes that finer comminution which more perfectly exposes its gold content.

In order to employ heavy gravitation stamps to full advantage it is absolutely necessary that a uniform speed should be maintained by proper governing of the engine. Only under these conditions can the maximum speed and height of drop be used without fear of camming; and the damage done in a few moments by sudden increased speed above the normal may prove a costly matter, both in loss of time and material. Further, with heavy stamps, the use of five-stamp camshafts is very desirable, in that, as compared with 10 or 15-stamp camshafts, both the power to be transmitted and the number of shocks due to lifting stamps are reduced by one-half or two-thirds for each shaft.

The future limit of stamp weight is difficult to foretell, but it will probably be determined more by the mechanical considerations involved, as in the cam system of lifting, than by any decreased relative efficiency as a device for pulverizing ore. The advantages of heavy stamps as compared with light ones may be briefly stated as follows:

1. Reduction of the initial capital expenditure in erecting, say, 200 stamps at 1750 lb. with accessories, in place of 280 stamps at 1250 lb. each.

2. Reduction of size in mill building, almost proportionate to the less number of stamps.

3. Thirty per cent less shafting, belting and other moving parts to maintain.

4. Thirty per cent less labor required for dresssing plates, lubricating moving parts, changing screens and other work incidental to milling operations.

* * *

The discussion of Dr. Caldecott's contribution has been very extended, and we append hereto a few extracts of comments by one of his critics, together with some extracts from the author's reply.

Mr. E. J. Way: While Dr. Caldecott is very emphatic in his personal opinion that the increase of weight already carried out does show very appreciable economic improvements, this opinion is not supported by any convincing arguments or array of experimentally proved data. I had been led to hope that the author's unrivalled opportunities for arriving at definite conclusions based on experimentally determined facts would have enabled him to throw some very clear light at last on that most important question: What are the limits within which primary and secondary crushing machines would operate at their best efficiency and give the best return for the money invested? I had hoped that this question would have been fully treated from every standpoint, namely, first cost, running and maintenance cost, life of mine, and all the other factors that enter into the question of economic efficiency in contradistinction to mere mechanical efficiency. The laudable and useful objective of the experiments seems to have been lost sight of, and the experiments to have been carried out merely to see how the capacity of the stamps could be increased. A careful examination of the data brought forward by the author on which he apparently based his recommendation that heavy stamps should be installed at the mines mentioned (a recommendation, by the way, hardly confirmed by the total working costs per ton milled at the mines at the present time) simply shows that, the number, height of drop and height of discharge remaining the same, the extra amount crushed is almost directly proportional to the weight of the stamps. In other words, there is no increase of efficiency, but only of capacity. A further examination shows that in order to obtain the increased tonnage it was necessary with an increase of 32 per cent in stamp weight to increase the water per ton of ore crushed 41 per cent, while the amount of +60-mesh product increased 126 per cent. This would appear to indicate that, so far from there being an increase in the economic efficiency due to crushing large tonnages through fine-mesh screening by heavy stamps, there is actually a decrease in economic efficiency.

The experiments on preliminary fine breaking with rolls before milling might with profit have been extended to trying fine breaking in conjunction with lighter stamps, lower drop and greater number of blows per minute, as it would certainly appear, theoretically, that for every size of feed there should be, in order to obtain a maximum mechanical efficiency, a corresponding pressure over the unit area resulting from the blow of the stamp and depending on the three factors, weight of stamp, area of shoe and height of drop. It would appear that for a fine feed a light stamp with a low drop and running at a high rate of speed would prove most efficient, and it is to be regretted that the author's experiments were not carried further in this direction.

The author states that "fine breaking before milling with heavy stamps does little good," and goes on to say that "the heavier the stamp the coarser the preliminary breaking admissible, and vice versa." In other words, the heavier the stamp the coarser the preliminary breaking admissible, and conversely, the lighter the stamp the finer the breaking admissible. I am of the opinion that here the author is getting at the very root

of the matter. I am inclined to believe that if we looked upon the heavy stamp as a fine breaker and were to interpolate it, in place of fine rolls, between the rock breaker proper and the light stamps, very appreciable gains in economy might result, as it must not be forgotten that the heavy stamp, even when crushing through four-mesh screening, produces as a by-product some 25 per cent of —90 grade. The fact that maximum mechanical efficiency when using heavy stamps is obtained only by crushing through correspondingly coarse screens, moves the sphere of action of heavy stamps more and more, with increase of weight, in the direction of coarse crushing. The economic limit of the weight of these stamps, leaving out mechanical difficulties, appears to be reached only when they encroach on the province of the jaw breaker or gyratory crusher with which they cannot compete. This economic limit will be reached when the mechanical advantages of the jaw breaker are balanced by the advantages accruing from the percentage of —90 grade produced by the heavy stamp, even when crushing through coarse screen.

As both heavy and light stamps, if properly designed for the size of ore fed and screen used, would be working at the maximum mechanical efficiency, and as the mechanical efficiency of the tube mill working under the best conditions is only about 25 to 33 per cent of that of a properly designed stamp mill, it would appear that both the mechanical and economical efficiencies of a plant thus designed should be considerably better than that of the present fashionable heavy stamp plus tube mill combination.

Commenting on the four advantages which Dr. Caldecott ascribes to installations of heavy stamps. I should like to make the following observations:

1. The reduction of capital expenditure on stamps and accessories is not as great as might appear probable at first sight, as the total weight of the stamps themselves is identical, while the greater vibration consequent upon the heavier weight of the stamps necessitates the use of massive concrete foundations in place of the old timber piles, the replacement of the wooden king post by steel or reinforced concrete construction, and the installation of far heavier and larger cams and camshafts, pulleys, friction clutches, etc. The total quantity of water passing through each box will be more than doubled, while the larger ore feed will necessitate bigger and more expensive feeders. If to the cost of the stamps be added the cost of the necessary secondary grinding plant, such as tube mills, with their launders, cones, pumps and other equipment, it will be obvious that the cost of crushing appliances alone will be higher with heavy stamps than with light ones.

2. As to there being any reduction in the cost of the mill building, this is more than doubtful if the mill bins are retained as an integral part of the building. The usual allowance of three days' storage capacity means that the old allowance of 12 to 15 tons per stamp will have to be increased to 60 tons per stamp, necessitating, in the case submitted by the author, a storage capacity of 12,000 tons, as against 4200 tons. This enormous increase would cause greatly increased strains throughout the bin structure, and would probably make the cost of mill buildings greater with heavy stamps than with light ones.

3. There will certainly be some 30 per cent fewer moving parts, but as these will all be larger and the wear and tear will be correspondingly greater, I very much doubt whether the maintenance and renewal costs in the mill alone will be greatly reduced, and if to these costs are to be added the tube mill maintenance costs, the total maintenance costs will almost certainly be higher.

4. I do not see that there can be as large a decrease as 30 per cent in the labor requirements. Take the question of dressing plates. It has been found that when crushing through coarse mesh screening it is necessary, owing to the scouring produced by the large particles and the increased grade of the plates required to allow the pulp to flow freely, to remove the plates from the mill building and place them below the tube mills. Here there will be no reason why their number should correspond with the number of mortar boxes in the mill, and, as a matter of fact, their number will probably be not less than in a light stamp mill of similar capacity, so that the labor of dressing plates will probably be the same in both cases. Any saving of labor in other work in the mill will probably be offset by the work in connection with the running of the tube mills. Generally throughout the plant costs will probably be found to be higher in two-stage than in single-stage crushing, due to, first, the heavier running and maintenance costs of the heavy stamps and tube mill; second, the absolute necessity of skilled supervision of the tube mill plant, if it is to run at its low maximum efficiency, and, third, the increased pumping costs in a heavy stamp and tubemill combination due to steeper grade of launders or to the increased volume of water to be handled. It must not be forgotten, too, that the smaller and more numerous the units, the less the effect of one or more units being out of action.

Dr. W. A. CALDECOTT in his reply to discussion noted the following points, among others:

The suggestion that crushing plants comprising both stamps and tubemills should be considered as a whole is, of course, correct, and this view has been usual on the Rand since the introduction of secondary crushing appliances in the form of tubemills. In this connection the author has found very convenient for purposes of calculation what may be termed a "nominal crushing unit" (N. C. U.) defined as a stamp with a running weight of 1250 lb. and an assumed new weight of 1350 lb. The number of drops per minute is assumed to be 100, the set height of drop 8 in., and the actual height of drop 7.5 in. A tubemill 5½ ft. x 22 ft. is assumed to be equal to 30 such N. C. U., and likewise requires about 100 hp to operate; this ratio of stamps to tubemill is not constant, but varies somewhat according to the conditions of operating. The capacity of the above nominal stamp is rated by the tons of —90-mesh product it yields per 24 working hours, and the capacity of stamps with different running weights, but operating under similar conditions is calculated in proportion to such different running weight, while the capacity of tubemills of other dimensions than those cited is taken in proportion to the cubic capacity of their shells. Thus a stamp with 1400 lb. running weight is equal to 1.12 N. C. U., and a tubemill 14 ft. long by 4 ft. diameter is equal to 10 N. C. U. This simple rule is of value in determining the crushing plant required when a final pulp containing a given percentage of —90-mesh product is desired. Thus, to estimate how many tubemills (5½ ft. x 22 ft.) must be added to a plant containing 200 stamps with a running weight of 1400 lb. in order that 1420 tons daily of pyritic banket ore should be crushed to a pulp containing 80 per cent —90-mesh product, and assuming likewise that an N. C. U. produces 3.3 tons —90-mesh product per 24 hours:

80 per cent of 1420 tons = 1136 tons —90 product.

200 stamps at 1400 lb. = 224 N. C. U.

224 N. C. U. × 3.3 tons = 739 tons —90 product.

$$\frac{1136 \text{ tons} - 739 \text{ tons}}{30 \times 3.3} = \text{four tubemills } (5\tfrac{1}{2} \text{ ft. x } 22 \text{ ft.})$$

The arrangement in the City Deep mill of multiple camshaft bearings with 2000 lb. stamps, which have an estimated 11-ton duty, constitutes a logical means of coping with the shock due to impact of cam with tappet, which naturally increase in severity with increased stamp weight. Whether some form of lifting device other than the cam will finally be adapted to the relatively small crushing unit which the present stamp constitutes it is difficult to say, or whether the encouraging results of recent coarse crushing trials, which are given later in this reply, will result in the use of large pneumatic or other stamps with coarse screening, and comparable in duty with the steam

stamps on Lake Superior. This last system, however, appears the most promising, and it is to be hoped that the pioneer work already carried out at the New Kleinfontein mill in this direction will be further developed. With the elimination of amalgamation in the battery, rolls to reduce ore to pass a screen of nine holes per square inch would produce a suitable product for tubemilling at a low capital cost, could steel be found for shells which would preserve true faces by withstanding abrasion well, and thus minimize stoppages and cost of truing the roll faces. The comparative advantage of the stamp in this respect is that even though the crushing surfaces of shoe and die wear irregularly, yet a fair crushing efficiency is maintained.

The author has recently carried out the following tests to further develop certain points referred to in the paper:

Five stamps with long heads and short stems were extemporized by joining two 24-in. heads together by means of stout pins, and were operated in a mortar box with the front cut out and replaced by a thin detachable steel plate. A wood and canvas housing prevented splash above the mortar box through which the heads projected. This design of stamp was on the lines of the early experiments carried out at the Knights Deep of increasing the weight of stamps by interposing a false shoe between the head and the true shoe, although the advantages of such a design were not then realized. The open front mortar box facilitated access to the head when required, and the long head combined with short stem lowered the center of gravity of the stamp to an appreciable and desirable extent as compared with the usual type. While the stamp in question was of a heavy type, obviously the design is applicable to stamps of any weight and has now been adopted for the new City Deep mill.

A still more important investigation conducted by Mr. W. R. Dowling, at the Simmer East mill, concerned the maximum size of particle which could be efficiently pulverized in the usual tubemill, 5½ ft. x 22 ft., or, in other words, the coarsest battery screen practicable under these conditions. Briefly, the maximum efficient aperture was found to be 0.27 in., equivalent to battery screen of nine holes per square inch, and besides obtaining very high stamp duties, the actual tonnage per day of the desired —90-mesh product of stamps and tubemills was further found to be much greater by using such coarse screening than with the finer battery screens in common use. The fact that pulp from a coarse battery screen should contain more tons of fine product per stamp than that from a fine screen would hardly be expected, but though the percentage of fine product in the former pulp is low, yet this is more than offset by the high stamp duty. The general improvement in crushing efficiency is really due to applying the principle of promptly removing from the mortar box all particles too small for the stamp to readily reduce further, and by feeding the coarser of these to the tubemill, providing the latter with material which is within the limits of its most efficient operation.

In the Simmer East investigations some trials of 50 stamps with coarse battery screening (16 and 36 holes per square inch) and the remaining 80 stamps with fine screening (1000 to 1600 holes per square inch) were first carried out with encouraging results, though the scouring of the amalgamated plates with this coarse product was found to be excessive, and in subsequent trials the plates were covered with linoleum. The cyaniding results of the trials were normal and in proportion to the ultimate fineness of the pulp.

In all trials the usual tubemill circuit was in operation whereby the coarse portion of the pulp after elevation was separated by cone classifiers for tubemilling. The tubemill outflow, after passing over shaking amalgamated plates, rejoined the pulp from the batteries for re-elevation and separation of any coarse particles, which again passed through the tubemills, while the fine material overflowing the classifiers gravitated to the cyanide plant.

In the subsequent trials, only stamps with coarse screening were used, and the tonnage of ordinary mill feed crushed was calculated from numerous measurements of ore supplied to the batteries from a bin, which was partitioned off for the purpose, and the surface of the ore levelled before and after each run, 20 cu. ft. of ore being taken to the ton. The duties thus found were checked where practicable and with substantially the same results, by the tonnage of sand and slime collected during each run. Fifty stamps and three tubemills were available for the trials and were run in carrying ratios to each other.

Trials were made with battery screen of four holes per square inch, but it was found that with this product the tubemills discharged very coarse material, which accumulated in the tubemill circuit, many particles in the pulp being apparently too large to be shattered by the falling pebbles, and being nearly rounded by abrasion during their passage through the tubemill.

Regular and frequent sampling of battery pulp during each run was carried out to obtain samples for grading analyses and to determine the ratio of water to solids. The grading of the final pulp was calculated back from the grading of the sand charges, in accordance with the percentage by weight they constituted of the ore milled. The production of —90-mesh product per stamp was determined from the percentage of —90 mesh in the battery pulp and the stamp duty, while the product of —90-mesh product per tubemill was calculated from the tonnage of ore milled during each run, divided by the number of tubemills in operation, and multiplied by the difference between the percentage of —90 product in the battery pulp and the final pulp.

The average running weight of the stamps in all trials was 1400 lb., with 8 in. set height of drop at 98 drops per minute, and 2 in. height of discharge, while all mortar boxes were fitted with back-water feed as well as the usual supply from above. The utility of the back-water feed in coarse crushing may be attributed to its washing toward the screen particles which are fine enough to pass through, though too coarse to readily become suspended in the splashed pulp.

The increased efficiency of tubemilling, as measured in tons of —90-mesh product, with coarse battery screening and a consequent coarse tubemill feed, may be explained as due to the fact that such coarse particles provide work to be done by the falling pebbles throughout the full length of the mill, whereas with a finer feed many particles are early reduced to the required size and hence the pebbles near the outlet of the mill are not supplied with sufficient coarse particles to secure their full crushing efficiency.

Oildag and Graphite.—The Acheson Oildag Company, of Niagara Falls, N. Y., has purchased a factory site of 30 acres at Port Huron, Mich., and will erect there their prospective plant. Port Huron was selected as the site because of its comparatively pure water and excellent shipping facilities. Oildag has made phenomenal progress in the matter of establishing itself as a superlative lubricant and conserver of petroleum lubricating oils, but owing to the past limited manufacturing capacity of the company, the price was of necessity high and retarded its rapid introduction into general lubrication. On the first of July, the Acheson Oildag Company reduced the selling price of Oildag to approximately 60 per cent of that previously quoted, thereby bringing it down to a price, when bought in quantity, of considerably less per gallon than that of the oil with which it is mixed. The presence of the deflocculated graphite in the oil makes each gallon equal to 2 gal. to 4 gal. of plain oil. The International Acheson Graphite Company has arranged to erect a very substantial and large addition to its works at Niagara Falls. This will afford capacity for 10 or 12 new electric furnaces. The buildings will be of brick and steel construction and erected to the west of its present works, where there is ample room for still further enlargement. The additional furnaces will make possible a very large increase in the output of graphite rendered necessary by the very rapidly increasing demand.

Analysis of Manganese Bronze.

BY H. B. SWAN.

The analysis of this well-known and important alloy necessitates the determination of tin, copper, iron, aluminium, manganese and zinc. Tin, iron, aluminium and manganese are present in relatively small amounts, and inasmuch as a difference of a few tenths of 1 per-cent of these ingredients may materially alter the physical properties of this bronze, it is extremely important at times to have accurate and reliable results. The method outlined below has been worked out in this laboratory; it has given excellent results and is comparatively rapid, a period of about six to seven hours being required for the complete analysis.

A 5-gram sample of drillings is weighed out into a No. 2 beaker, 20 c.c. of water added, and then carefully 20 c.c. of HNO_3 (specific gravity 1.42). Cover, and after the reaction has subsided, heat to complete solution, and boil for about five minutes. This strength of acid has been found most satisfactory, as in using a more dilute solvent the precipitate of metastannic acid runs through even a double filter.

Filter off the precipitate of metastannic acid through a double filter of "S & S" 589 white ribbon paper, using a platinum cone and employing suction. Wash the precipitate well with half-and-half nitric and allow the suction to act sufficiently long after the paper is drained to dry it somewhat. Ignite the precipitate and paper together, charring carefully, and afterwards raising the heat to the full heat of a Bunsen or Tyrell burner. Calculate the tin from the precipitate of SnO_2. It is not necessary to subject this precipitate to any further treatment to purify it, as it is so small that only traces of iron or manganese can be precipitated along with the tin.

To the filtrate is added 5 c.c. of concentrated H_2SO_4 (sp. g. 1.84) and the whole evaporated on the hot plate or bath to almost gentle dryness. That is to free it from the nitric acid, and occasionally, if a low-grade spelter is used in the manufacture of the alloy, to precipitate any lead present.

If there is an appreciable amount of lead present and it is desired to determine it, after cooling the crust of sulphates in the bottom of the casserole, about 50 c.c. of water are added and the sulphates will readily dissolve with a little heating. The lead sulphate is filtered off on to an asbestos pad in a platinum gooch crucible, washed with a 5 per cent solution of H_2SO_4, then, after removing the filtrate, washed with alcohol into a clean flask. The crucible is then ignited at a low heat, but sufficient to remove any H_2SO_4 that may still be present.

The filtrate is then transferred to a 500 c.c. graduated flask and diluted to 500 c.c.

For copper either pipette off 100 c.c. or measure out, by means of a 100-c.c. graduated flask, 100 c.c. of the solution, and put into a tall non-lipped 150-c.c. beaker. Just neutralize with NH_4OH and then add about 3 c.c. of a mixture of three parts nitric (sp. g. 1.42) to one of sulphuric acid (sp. g. 1.84) and submit the solution to electrolysis. This laboratory uses a gauze cathode and a rotating anode, and employs a current of $N.D._{100} = \frac{3}{4}$ amp. After testing with H_2S on a spot plate to insure the complete removal of the copper, the cathode is washed, as it is removed from the solution and after removing, with H_2O, and then dipped in alcohol and dried. A quick and very satisfactory method of drying consists in holding the cathode above a piece of asbestos, covering the top of a Tyrell burner, and at the same time allowing a current of compressed air (not too strong) to play over the gauze.

Two 100-c.c. portions of the solution are run simultaneously, and, after the removal of the copper, iron is determined in one portion and aluminium in the other. To one portion add 5 grams of ammonium chloride and make just slightly alkaline with NH_4OH and heat to the boiling point. Cool, allow to settle and filter. Iron and aluminium are precipitated. Wash with dilute ammonium chloride solution. The precipitate is redissolved in 5 c.c. of HCl (1.12) and the solution diluted up to 400 c.c. in volume. Add 10 c.c. of a 10 per cent solution of sodium ammonium hydrogen phosphate and then NH_4OH till just slight but permanent precipitate forms. Then add 1½ c.c. of strong HCl and then 10-15 c.c. of a 20 per cent solution of sodium thio sulphate. Heat to boiling, and after the boiling point is reached add a mixture of 15 c.c. of a 20 per cent solution of ammonium and 8 c.c. of strong acetic acid. Boil for about ten minutes, all to settle, and filter. The precipitate filters readily, but is hastened by using suction. Wash several times with hot water. The paper and precipitate are burned together and then ignited over the blast lamp, either in a porcelain or platinum crucible. Weigh as $Al\ PO_4$, the conversion factor being 0.2219 for aluminium.

The iron, aluminium and manganese are precipitated together in the other 100-c.c. portion with NH_4OH. The precipitate redissolved in about 5 c.c. of strong HCl. The iron reduced with stannous chloride and then titrated with a solution of $\frac{N}{80} K\ Mn\ O_4$, using a guard solution made by mixing phosphoric acid, manganese sulphate, and sulphuric acid together. For details of this method see Blair's methods for the analysis of iron and steel.

Now two other 100-c.c. portions of the solution are used for the determination of manganese and zinc. The copper may be removed either with metallic aluminium or by electrolysis. In the latter case the copper is removed from the sulphuric acid solution without the addition of ammonia or nitric acid, but adding about 1 c.c. of H_2SO_4 for a total volume of 125 c.c. of solution.

After removing copper, for manganese add a few c.c. of strong bromine water and make ammoniacal and then add a slight excess. Heat till all the iron, aluminium and manganese are precipitated, then filter, wash and dissolve in a few c.c. of sulphurous acid. Now add to the solution 10 c.c. of HNO_3 acid and 30 c.c. of H_2O and boil a minute, then add about ¼ gram of sodium bismuthate and heat till the pink color completely disappears. Add a few drops of sulphurous acid to clear the solution and boil again to expel all nitrous fumes. Cool the solution till quite cold and add a slight excess of sodium bismuthate again. Agitate a minute or two, then add 50 c.c. of 3 per cent HNO_3 and filter through asbestos on a platinum cone, using suction. Wash with 50 c.c. more of 3 per cent HNO_3 and then reduce with ferrous ammonium sulphate and titrate with a solution of $K\ Mn\ O_4$, which has been standardized against the ferrous ammonium sulphate solution.

This is merely an application of the well-known bismuthate method for very small amounts of manganese and the details may be found in any standard manual for volumetric work. It is said to be the most accurate method for manganese determinations in which the manganese does not exceed 2 per cent. In manganese bronze the manganese does not usually exceed 0.25 per cent.

Zinc is determined in the other c.c. portion of the solution from which the copper has been removed, without the addition of nitric acid. Zinc may be estimated either as the pyrophosphate, which is the more accurate method, or by electrolysis, which is the more rapid, but not quite so accurate. When selecting the pyrophosphate method, separate the zinc by hydrogen sulphide from a slightly acid solution of formic acid. This is done by neutralizing the solution with Na_2CO_3 and then adding ½ c.c. of 50 per cent formic acid. Now run in H_2S till all the zinc is precipitated.

Zinc sulphide is precipitated free from iron, manganese, and aluminium, and can be filtered quickly and completely in a short time by using asbestos on a platinum cone and employing suction. The ZnS is then dissolved in HCl and precipitated as the phosphate with $NaNH_4HPO_4$.

If the electrolytic method is selected, the zinc is plated out from the solution made sufficiently alkaline to redissolve the precipitated zinc and aluminium. The iron which is precipitated along with the zinc and aluminium by the sodium hydrate

is filtered off before the electrolysis is proceeded with. It is best to boil the alkaline solution to free the iron hydrate from any zinc. The filtrate from the iron is then cooled and the zinc plated out on a copper-plated cathode, with the use of a rotating anode.

By the use of the method outlined above we repeatedly obtain results which check closely and total from 99.85 per cent to 99.99 per cent.

Lumen Bearing Company,
Buffalo, N. Y.

The Chemical Export Trade of Germany.

By W. Pollard Digby.

Of all national manufacturing industries the chemical trade of Germany would probably be selected by an impartial observer as the most characteristic of Teutonic application and adaptiveness. Her racial attributes of co-ordination, disciplined organization, and painstaking research, no less than her splendid educational systems, have given Germany her just pride of place in this branch of the industrial arts.

Students of German conditions are apt to urge that her military system has dwarfed her initiative, and that for the original pioneer work in engineering, in metallurgy and in chemistry, we must look to nations like England, France or Sweden, whose natural initiative and whose preliminary discoveries have been the inspiration upon the lines of which the less brilliant but more patient Teuton has plodded along and has achieved preeminence.

Even though the charge of lack of initiative has, in the writer's opinion, still some basis, there has to be set against this the advantage of an imagination which is educated, logical and scientific, which seizing on any vantage ground advances from it in an ordered manner to a definite end. One cannot, indeed, speak or even think of German chemical industry without thinking of the aniline dye industry, founded on British neglect of Perkin's discoveries, and of the success of the synthetic indigo industry with its dire effect on Indian indigo planters.

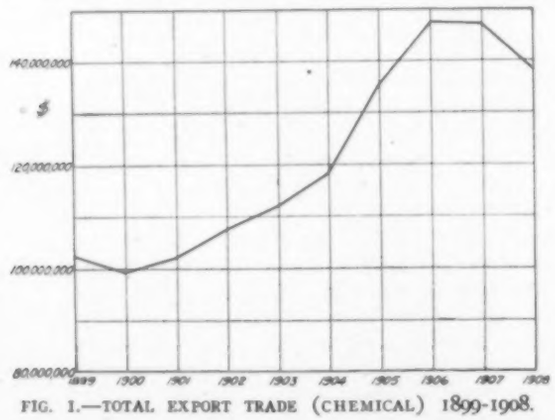


FIG. I.—TOTAL EXPORT TRADE (CHEMICAL) 1899-1908.

Co-ordination may be said to be the keynote of German success. It requires the best type of specialist to take the fruits of one branch of scientific research, and, securing them in another field, to establish yet fresh industries. The present position of the incandescent lamp may be said to be due to a harking back along the road traversed, and attacking with new methods and new material obstacles formerly regarded as inseparable. At first a matter of metallic filaments in the shape of platinum wire, then a sphere occupied wholly by the carbon-filament lamp. But where other countries were concerned in developing the latter, the Teuton scientist, using the rarer metals, discovered or cheapened by the discoveries in metallurgical chemistry the metallic-filament lamps.

Co-ordination has, moreover, not been wholly educational and scientific. Systematized transport has done much for the development both of internal industries and of external trade. Sociological conditions, to which it is beyond the scope of this article to allude, have also contributed.

As in England and in the United States, local industries have been intimately associated with the chemical trade, and Germany's advances in the textile, porcelain and paper industries have profited largely by the advances made in chemistry.

Taking the 10 years, 1899 to 1908, as in previous articles, we find that the export chemical trade reached a maximum of $147,551,000 in 1906 and a minimum of $99,404,000 in 1900, the average being $121,077,000.

Fig. 1 shows the annual fluctuations in the total trade, while the following table shows the percentage ratio between the chemical and total exports, as well as the estimated population for each year and the value of the chemical exports per head.

	1899.	1900.	1901.	1902.	1903.
Chemical exports	$102,467,000	$99,404,000	$102,092,000	$107,603,000	$112,061,000
Total exports	1,051,750,000	1,152,850,000	1,107,850,000	1,169,450,000	1,253,650,000
Percentage ratio	9.75	8.62	9.23	9.28	8.94
	1904.	1905.	1906.	1907.	1908.
Chemical exports	$118,375,000	$135,585,000	$147,551,000	$147,113,000	$138,514,000
Total exports	1,305,650,000	1,432,900,000	1,589,750,000	1,711,300,000	1,599,625,000
Percentage ratio	9.07	9.45	9.28	8.60	8.66

The general tendency of the first set of figures is to decrease, while the latter shows a more upward tendency.

	1899.	1900.	1901.	1905.
Population	55,248,000	56,367,178	56,874,000	57,767,000
Value of chemical exports per head	1.86	1.77	1.80	1.87
	1903.	1904.	1905.	1906.
Population	58,629,000	59,475,000	60,641,278	61,177,000
Value of chemical exports per head	1.91	1.99	2.24	2.41
			1907.	1908.
Population			62,083,000	62,982,000
Value of exports per head			2.37	2.20

In the first of these articles it was pointed out how the prosperity or otherwise of the chemical industry depended to a very large extent on the activity or stagnation of the other industries which are directly or indirectly connected with it, and hence it is that the rise and fall of the figures for the total chemical output affords, like the market prices of copper, a very close guide to the general trend of commercial activity, and the marked fall in the export figures for the years 1907 and 1908 correspond with the general depression in the industrial world which has characterized the latter part of last decade.

The total figures given above include, as before, drugs and dyes, as well as chemicals in the stricter sense of the word.

Turning now to a more detailed examination of the most important chemicals, the exports have as before, apportioned to the six zones: Great Britain, rest of Europe, United States, British North America, other British Colonies, South America, and the rest of the world, as affording a convenient and instructive method of subdivision.

Figures for **Sulphuric Acid** are low—Great Britain and the United States, the chief consumers, making their own in large and increasing quantities, require none from Germany.

TABLE I.—GERMAN CHEMICAL EXPORTS.

Sulphuric Acid.	1904.	1905.	1906.	1907.	1908.
Great Britain.....					
Rest of Europe.....	$500,000	$450,000	$561,250	$566,500	$708,000
United States.....					
Brit. North America..					
Other Brit. Colonies..					
South America					
Rest of World.....	100,000	100,000	92,250	57,250	49,250
Total	$600,000	$550,000	$653,500	$623,750	$757,250

Strassfurt (Saline) Deposit.—This, an export trade in raw material rather than in a finished product, is a large and growing industry, the United States being the best customers. Separate enumeration was not made previous to the year 1906.

TABLE II.

Strassfurt (Saline) Deposit.	1906.	1907.	1908.
Great Britain	$388,750	$441,000	$405,000
Rest of Europe	1,347,250	971,500	1,891,000
United States	1,994,500	2,112,000	1,869,250
British North America			
Other colonies			13,000
South America			
Rest of World	37,250	780,000	22,000
Total	$3,767,750	$4,304,500	$4,200,250

Caustic Soda, Soda Ash and Caustic Potash.—Great Britain is the chief producer of caustic soda, and as far as soda ash is concerned "Rest of Europe" accounts for the major portion of the German export trade.

For potassium salts Germany is preëminent, this being the one point in which Germany has greater natural advantages than England in the shape of mineral deposits necessary for this industry. The detailed tables of figures for these chemicals are set out below:

TABLE III.

Caustic Soda.	1904.	1905.	1906.	1907.	1908.
Great Britain					
Rest of Europe	$175,000	$175,000	$247,750	$340,250	$353,000
United States					
Br. Nth. America					
Other B. Colonies.					
South America					
Rest of World	100,000	125,000	68,000	44,500	60,250
Total	$275,000	$300,000	$315,750	$384,750	$413,250

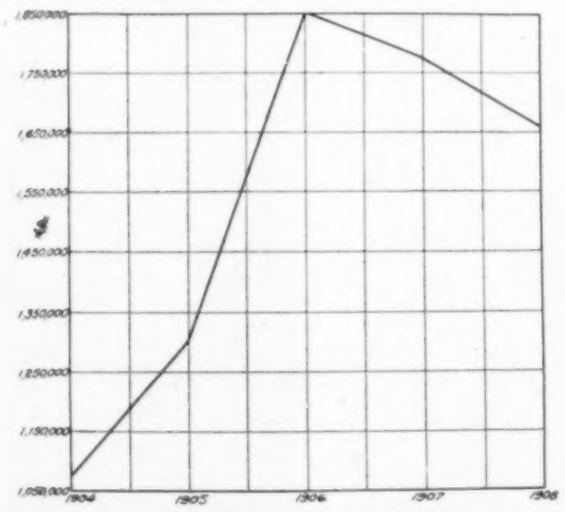


FIG. 2.—EXPORT TRADE IN POTASSIUM AND SODIUM CYANIDE (1904-1908).

TABLE IV.

Soda Ash.					
Great Britain	$25 000	$25,000			
Rest of Europe	900,000	900,000	$830,000	$757,000	$1,268,250
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	50,000	75,000	56,250	27,500	12,500
Total	$975 000	$1,000,000	$886,250	$784,500	$1,280,750

TABLE V.

Caustic Potash.					
Great Britain	$450,000	$300,000	$334,000	$382,500	$323,250
Rest of Europe	1,525,000	1,475,000	1,394,750	1,201,500	1,698 750
United States	125,000	100,000	132,500	154,500	143,000
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	75,000	75,000	43,250	34,500	27,750
Total	$2.175,000	$1,950,000	$1,905,000	$1,773,000	$2,192,750

Bleaching Powder.—These figures show a down-grade tendency; it is quite possible that the success of the Castner-Kellner process in England may partly account for this, as the demand for this product is still as great, if not greater, than ever.

TABLE VI.

Bleaching Powder.	1904.	1905.	1906.	1907.	1908.
Great Britain	$250,000	$275,000	$253,000	$164,250	$148.750
Rest of Europe	100,000	125,000	167,000	163,250	188 250
United States	175,000	225,000	183,750	175,750	142,500
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	150.000	75,000	62,000	34,250	58,750
Total	$675 000	$700,000	$665,750	$537,500	$538,250

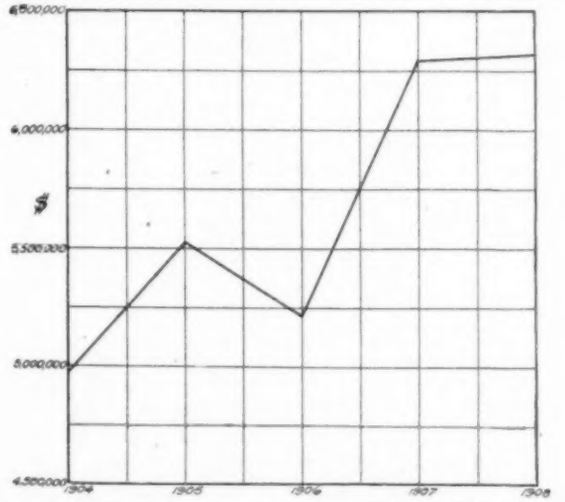


FIG. 3.—EXPORT TRADE IN POTASSIUM CHLORIDE (1904-1908).

Sulphate of Copper.—Here again the export trade is not large, that to Great Britain being too small to merit enumeration. The value for the five years under notice is as follows:

TABLE VII.

Sulphate of Copper.	1904.	1905.	1906.	1907.	1908.
Great Britain					
Rest of Europe	$175,000	$125,000	$333,500	$251,000	$271,500
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	50,000	75,000	60,750	44,250	44.000
Total	$225,000	$200,000	$394,250	$295,250	$315 500

Sodium and Potassium Nitrate.—In the case of these two chemicals we have an example of a large and increasingly important industry. In the case of the sodium salt, the export to Great Britain is not large, but the trade to the United Kingdom in potassium nitrate is considerable, while the total trade, as the following table shows, has more than doubled since 1904.

TABLE VIII.

Sodium Nitrate.	1904.	1905.	1906.	1907.	1908.
Great Britain			$18,000		$39.750
Rest of Europe	$925,000	$1,000,000	1,121,250	$1.481,000	1,301.250
United States					
Br. North America.					
Other B. Colonies.					
South America					
Rest of World	125,000	75,000	40,750	69,000	74,250
Total	$1,050.000	$1,075,000	$1,184,000	$1.550,000	$1,415,250

TABLE IX.

Potassium Nitrate.	1904.	1905.	1906.	1907.	1908.
Great Britain	$350,000	$400,000	$354,500	$578,500	$488,250
Rest of Europe	450,000	525,000	461,250	511,250	365,000
United States					
Br. North America.			14,000	23,250	42.250
Other B. Colonies.			21,000		
South America	50,000	50,000	72,500	77,250	78,000
Rest of World	250,000	325,000	238,250	240,000	201,250
Total	$1,100 000	$1,300,000	$1,161,500	$1,480,250	$1,174,750

Carbonate of Potash is an industry of increasing importance. The United States trade accounts for about one-half of the total exports.

TABLE X.

Carbonate of Potash.	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$200,000	$200,000	$206,250	$228,500	$222,500
Rest of Europe...	175,000	200,000	236,250	331,000	356,500
United States.....	400,000	525,000	543,000	611,500	522,750
Br. North America.					
Other B. Colonies.					
South America....					
Rest of World....	175,000	125,000	90,750	70,750	64,500
Total	$950,000	$1,050,000	$1,076,250	$1,241,750	$1,162,250

Sodium Sulphate and Potassium Sulphate.—Separate enumeration of the latter was not given previous to 1906. The large trade in the latter salt with the United States is worth noticing.

TABLE XI.

Sodium Sulphate.	1904.	1905.	1906.	1907.	1908.
Great Britain.....					$32,000
Rest of Europe...	$250,000	$325,000	$436,500	$453,500	518,500
United States.....					
Br. North America.					
Other B. Colonies.					
South America....					
Rest of World....	50,000	50,000	45,250	66,000	38,250
Total	$300,000	$375,000	$481,750	$539,500	$588,750

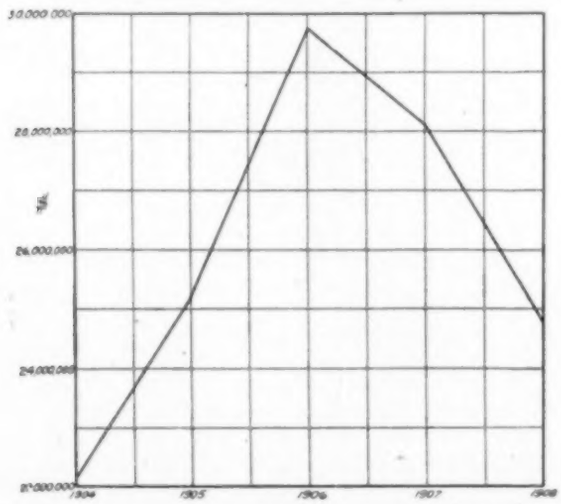


FIG. 4.—EXPORT TRADE IN ANILINE DYES (1904-1908).

TABLE XII.

Potassium Sulphate.			
Great Britain.....	$148,500	$238,500	$588,750
Rest of Europe.....	312,500	462,500	495,000
United States.....	1,045,500	988,750	1,073,000
British North America.....			
Other British Colonies.....	31,000	92,250	45,750
South America		32,750	13,750
Rest of World.....	126,000	114,250	198,000
Total	$1,663,500	$1,929,000	$1,914,250

Potassium and Sodium Cyanide.—There is, of course, a large trade in these products to Africa and other mining regions, the total export value amounting in 1906 to nearly $2,000,000. A curve is given in Fig. 2 of the total trade for the years 1904-1908.

TABLE XIII.

Potassium and Sodium Cyanide.	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$50,000	$25,000	$47,500	$51,250	$52,250
Rest of Europe...	50,000	50,000	99,000	46,500	76,250
United States.....	225,000	300,000	458,000	488,000	491,000
Br. North America.				20,250	
Other B. Colonies.	525,000	775,000	1,067,000	1,019,750	924,750
South America....					
Rest of World....	225,000	150,000	180,250	197,750	115,750
Total	$1,075,000	$1,300,000	$1,851,750	$1,773,500	$1,660,000

Potassium Chloride.—This, as a necessary re-agent in the manufacture of potassium chlorate, is a very important and growing export product, as the following and Curve No. 3 shows the value of the total trade in 1908 annuity to as much as $6,320,750.

TABLE XIV.

Potassium Chloride.	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$450,000	$550,000	$459,250	$491,500	$419,250
Rest of Europe...	1,550,000	1,750,000	819,000	1,956,750	2,183,750
United States.....	2,975,000	3,100,000	3,871,250	3,792,500	3,646,250
Br. North America.					
Other B. Colonies.			24,250	30,250	29,250
South America....					
Rest of World....		125,000	44,750	23,500	42,250
Total	$4,975,000	$5,525,000	$5,218,500	$6,294,500	$6,320,750

Aniline Dyes and Indigo.—The trade in aniline dyes is, of course, the crowning triumph of the German chemical industry. Thirty-eight millions of dollars is indeed a figure to be justly proud of as the total value of one year's export. Figures for the last two years have fallen slightly, and there is the probability that the new English patent laws will not be without effect in lowering the value of the German exports to the United Kingdom. The distribution is shown in the following tables and a curve shows the fluctuation in the totals.

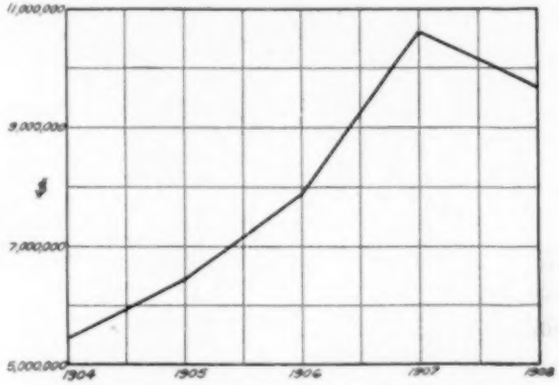


FIG. 5.—EXPORT TRADE IN INDIGO (1904-1908).

TABLE XV.

Aniline Dyes.	1904.	1905.	1906.	1907.	1908.
Great Britain..	$4,125,000	$4,525,000	$4,980,000	$4,978,250	$4,321,250
United States..	4,575,000	5,050,000	6,055,250	6,537,750	5,244,250
Rest of Europe.	6,850,000	10,175,000	11,547,500	9,778,250	9,917,250
Br. N. America.	75,000	200,000	138,000	139,000	111,500
Other Br. Col.	1,175,000	1,325,000	1,342,500	1,354,000	1,459,250
South America.	625,000	700,000	536,000	637,000	570,250
Rest of World.	4,725,000	3,200,000	5,162,000	4,688,750	3,149,750
Total	$22,150,000	$25,175,000	$29,761,250	$28,113,000	$24,773,500

TABLE XVI.

Indigo.					
United Kingdom.	$475,000	$625,000	$645,000	$513,250	$668,250
Rest of Europe.	2,350,000	2,225,000	2,549,000	2,899,500	2,596,500
United States..	875,000	1,025,000	1,126,500	1,363,000	1,182,750
Br. N. America.			17,750	28,000	14,250
Other Br. Col.	25,000	100,000	100,250	140,250	97,000
South America.			23,000		
Rest of Europe.	1,700,000	2,450,000	3,432,250	5,701,500	5,105,000
Total $5,425,000		$6,425,000	$7,893,750	$10,645,500	$9,663,750

Finally, a table is added showing the total distribution of chemical exports, from which it will be seen to what an extent Germany depends on Europe for its trade, Canada and South America consuming almost a negligible amount in proportion.

Total Distribution.	1904.	1905.	1906.	1907.	1908.
Great Britain.....	$6,375,000	$6,925,000	$7,834,750	$8,067,500	$7,709,250
Rest of Europe...	15,975,000	18,500,000	22,463,750	22,170,750	22,631,000
United States.....	9,350,000	10,325,000	14,283,750	14,860,750	13,133,000
Br. North America.			14,000	43,500	41,250
Other B. Colonies.	725,000	2,200,000	2,586,000	2,636,500	2,360,000
South America....	675,000	750,000	631,500	747,000	667,000
Rest of World....	1,350,000	1,375,000	1,185,750	1,797,500	1,008,750

Silver Shipments.—The output of the Rio Plata Mine in the Arteaga district of Chihuahua, Mexico, last month, was 55,779 oz. of silver. Nine hundred and forty-six tons of ore were treated. Sixty-eight bars, containing 70,000 oz. of silver, have been shipped to the city of Chihuahua since the first of July.

The Outcome of the Calcium Carbide Suits.

In our September issue, 1909 (Vol. VII, page 379), we noticed the decision of the United States Circuit Court, Northern District of New York, in two suits of the Union Carbide Company versus the American Carbide Company. One was based on the calcium carbide process patent of Thomas L. Wilson, the other on his product patent. Both decisions were in favor of the American Carbide Company, but the Union Carbide Company appealed from the decree of the lower court to the United States Circuit Court of Appeals. The suits have been tried before Judges Lacombe, Coxe and Noyes, and in the suit which is by far the most important one (the product suit) the decree of the lower court has been reversed and the Union Carbide Company has come out victorious.

The decision is written by Judge Noyes. The claim of the patent in question reads: "As a new product, crystalline calcium carbide, existing as masses of aggregated crystals, substantially as described."

The first defence of the defendant was that even if the patent broadly covers crystalline calcium carbide and even if all the calcium carbide of the prior art were amorphous, still there would be no patentable novelty in the crystalline form of calcium carbide; that the patent is not for a new product, but for a new form of an old product, having the same composition, properties and uses as the old.

In viewing the alleged anticipation by Woehler the decision of the court states that the Woehler product was the highest development of the prior art, but it was a black pulverulent mass, worthless commercially and never commercially used. "It would be unfit for use in gas generators and, we are satisfied, would rapidly deteriorate when exposed to the air. The product of the patent is more durable. It is hard, compact and so unlike the powdery mass as almost to amount to a new body. Moreover, we think the complainant correct in its contention that the iridescent surfaces of the crystalline carbide would protect it to some—although, perhaps, slight—extent from atmospheric action.

"It is also quite clear that Woehler published a mere result of a laboratory experiment which was put to no practical use. Crystalline carbide, on the other hand, has been a great commercial success and has furnished the foundation for important industries. . . .

"For these reasons it is held that the product of the patent possesses the requisite patentable novelty. And we must regard this conclusion as not only well founded in law, but as most just. To hold an important discovery which has given to the world a commercially new product—a product the high utility of which must be conceded—not entitled to protection for want of novelty would, as it seems to us, be applying the patent statute to defeat its fundamental purpose."

The defendant's second defence was that the patent, if broadly for crystalline calcium carbide, is invalid, because it is anticipated by the Woehler carbide. The question is whether the defendant has established the crystallinity of the article made by the Woehler process. "Upon this question the testimony of the expert witnesses called by the defendant and complainant is wholly irreconcilable. . . . It would serve no useful purpose to review this conflicting testimony or to discuss the reasons pro and con given by the experts. It is sufficient to say that upon careful consideration of the testimony we are not satisfied that the carbide of the experiment was crystalline. The burden is upon the defendant to establish crystallinity, not upon the complainant to disprove it. . . . Consequently, the Woehler product, whether correctly represented by the earlier compound or by the result of the recent experiments, must be held not to anticipate."

The defendant's third defence was that the patent was invalid because the claimed product was in public use for more than two years prior to the filing of the application. The court decides that while there were experimental uses of the product before that time, yet the proof fails to establish that there was any public use of the invention.

The fourth and last defence was that the defendant does not infringe. The file wrapper of the patent shows that Willson's original claim was as follows: "As a new product, crystalline calcium carbide having a bluish iridescence, substantially as described." This claim was rejected and the claim finally allowed by the Patent Office reads: "As a new product, crystalline calcium carbide existing as masses of aggregate crystals."

The decision of the court as to the meaning of the change of the claims states: "It is said that the applicant in his original application having broadly claimed crystalline calcium carbide and having acquiesced in that rejection of the claim cannot now contend that 'crystalline calcium carbide existing as masses of aggregated crystals' means nothing more than crystalline carbide. And we think this contention correct. The claim is undoubtedly limited to the form of crystalline carbide which exists as masses of aggregated crystals. There must be an aggregation. The claim does not cover an individual crystal nor any number of separate crystals not aggregated or joined together. To illustrate: had there been a question whether the Woehler powdery substance was amorphous or composed of minute, separate and distinct crystals, the claim as amended would have excluded it in either case, while the original claim would have been anticipated by it had it been composed of crystals."

As to the argument of the defence that the limitations of the word "crystalline" is not by the word "aggregated," but by the word "crystals" and that the term "masses of aggregated crystals" is used in a specific and limited sense, the court says: "We are by no means satisfied that any such fine distinction as this—based upon academic definitions in the Williams book—is so generally recognized." . . . "To confine the claim of the patent to masses of perfectly developed crystals would be, we think, by a strained construction to read out of the claim the only product producible by the process of the patent and to deny protection to a meritorious invention.

"In our opinion the patent covers crystalline carbide when the crystals are aggregated in masses, whether such crystals be perfect or imperfect, and as it is admitted that there are crystals in the defendant's carbide, and as those crystals are so aggregated, we think that the product of the defendant infringes.

"The decree of the Circuit Court is reversed with costs and the cause remanded with instructions to enter a decree for the complainant for an injunction, an accounting and costs."

The patent thus upheld by the Court of Appeals is No. 541,138, granted on June 18, 1895, to Thomas L. Willson and assigned to the Union Carbide Company.

* * *

In the second suit, the "process suit," the decision of Judges Lacombe, Coxe and Noyes reads as follows: "The question of infringement in this case depends upon the result of the inquiry whether the defendant employs the arc, as distinguished from the incandescent, principle in its electric furnace in the manufacture of calcium carbide. The burden is upon the complainant to establish by a fair preponderance of testimony—and not by mere scientific speculation—that the defendant does use the arc. We are, however, not entirely satisfied that the complainant has sustained this burden and accordingly feel constrained to affirm, with costs, the decree of the Circuit Court upon the opinion of the judge holding it, and it is so ordered."

Heating of cyanide solutions may be a distinct advantage in increasing the extraction of gold during the agitation of the pulp. A temperature of 90° F. is about the maximum limit. This may be accomplished by steam coils in the tanks, or by the introduction of live steam. At the Montana-Tonopah mill, Tonopah, Nevada, the extraction has been increased 5 per cent by this means, without any additional consumption of cyanide. The method seems occasionally worth considering and trying, especially where waste steam is available.

Hardness Tests on Alloys of Nickel and Copper with Electrolytic Iron—I.

By G. A. Roush.

While hardness as a *prime* factor is limited mainly to tool steels, it is still of sufficiently great importance in so many other materials that any investigation of this property is of interest. The relative hardness of any material varies with the kind of stress to which it is subjected. Hence different methods of measuring hardness, employing different methods of applying the testing stress will give results which are not strictly comparable. A series of samples arranged according to increasing hardness as measured by machining in a lathe will not necessarily remain in the same order when tested by a compression method. For this reason it is desirable that in any particular test the stress applied may be as nearly as possible similar to the stress to which the material will be subjected in use. In this investigation, however, no one particular use of the materials in hand was under consideration, so any method by which a series of comparative results could be obtained was sufficient.

The method used for determining the hardness was the scleroscope method devised by Albert F. Shore in 1906. The scleroscope was first put on the market in 1907, and is now used quite extensively in the commercial field as a means of determining hardness. The instrument consists of a vertical glass tube provided with a graduated scale suspended over a steel anvil, on which the sample to be tested is placed; a small

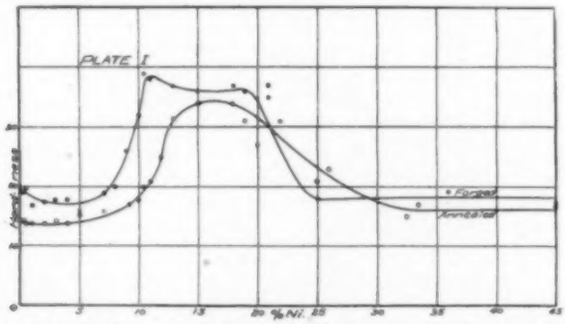


FIG. I.—HARDNESS OF FORGED AND ANNEALED NICKEL ALLOYS.

steel plunger with a diamond point, suspended in the top of the tube, is allowed to drop on the sample clamped on the anvil, the rebound of the plunger being read off on the scale at the back of the tube. When the plunger reaches the surface of the metal under test it carries a certain amount of potential energy, and the weight of the plunger, the height of the fall and the area of striking surface are so proportioned that in all cases this energy is sufficient to exceed the elastic limit of any material which may be under test. This being the case, the material on receiving the impact of the plunger is stressed beyond its elastic limit and given a permanent deformation. If the elastic limit of the material under test is high, the excess energy of the plunger is small, and only a small dent is made on its surface. That portion of the material immediately adjoining the deformed portion has then been stressed only up to its elastic limit, and in regaining its original form causes the plunger to rebound a certain distance. If the metal under test has a low elastic limit a smaller portion of the energy of the falling plunger is used in overcoming the elasticity, and a greater portion is used in displacing metal. The stress of the adjoining metal is then only sufficient to give the plunger a small rebound. This rebound is then taken to represent the hardness of the material under test.

Basing his conclusions on this line of reasoning, Shore makes the statement[1] that the curve for hardness, as determined by the scleroscope, is very nearly parallel to the curve of elasticity, but since the value obtained in any test for hardness depends to a certain extent on the method used to obtain the result, there is liability for considerable variation.

[1] *Amer. Mach.*, Nov. 14, 1907.

Preparation of the Materials.

This investigation covers a series of alloys of nickel and iron up to a nickel content of 50 per cent, and of copper and iron up to a copper content of 8 per cent, being a part of the extended study of electrolytic iron and iron alloys that is being carried on in the Chemical Engineering Laboratories of the University of Wisconsin, under grant from the Carnegie Institute. The results are of particular value in that they are obtained from materials the history of which is completely known, and which have been subjected to a number of other tests.[2]

The preparation of the electrolytic iron is sufficiently familiar that it does not require particular notice here. The iron used in the preparation of the alloys, according to several analyses, showed a purity of about 99.97 per cent, the latest analysis being as follows:

Carbon	0.000
Sulphur	0.003
Phosphorus	0.003
Silicon	0.002 or less.
Manganese	0.020
Iron, by difference	99.972 per cent.

The alloys were prepared by melting the iron with the calculated mounts of the added substance in magnesia crucibles in an electric resistance furnace, as described in the papers mentioned above. The amount of carbon absorbed in the melting has been shown by several different analyses to be less than 0.10 per cent, so that the hardening observed can be accounted as due to the addition elements.

The number of bars tested of each composition varies from 1 or 2 up to 12 or 15, and in each case 10 or more readings were taken on each bar, and averaged for the final result. Considerable variation was noticed in the various bars and in different parts of the same bar, due to irregularities in forging and annealing, but the general averages show a fair degree of uniformity.

Not all of the finished alloys were subjected to analysis, but a sufficiently large number of both the nickel and the copper series was analyzed to make one safe in making the assertion that all were sufficiently close to the calculated composition for the requirements of an investigation of this kind. A few analyses typical of the series are given below.

Nickel Alloys.

Mark.	Calculated.	Found.
144C	0.25	0.27
144F	2.0	1.93
144J	7.0	7.05
157H	12.0	12.07
166G	22.0	22.11
166L	35.0	35.09

Copper Alloys.

Mark.	Calculated.	Found.
158A	0.2	0.202
147H	1.0	1.006
158I	4.0	3.990
147U	7.0	7.050

All samples were tested in the forged condition and after annealing at 900° C.

Nickel-Iron Alloys.

There are no investigations described in the literature which are strictly comparable to the results given here, since in other

[2] See a, Physical Properties of Iron-Copper Alloys, Burgess and Aston Trans. Amer. Electrochem. Soc., 16, 241-256, 1909.
b, The Magnetic and Electrical Properties of Iron-Nickel Alloys, Burgess and Aston, Met. and Chem. Eng., 8, 23-26, 1910.
c, The Magnetic and Electrical Properties of Iron-Copper Alloys, Burgess and Aston, Met. and Chem. Eng., 8, 79-81, 1910.
d, The Strength of the Alloys of Nickel and Copper with Electrolytic Iron, Burgess and Aston, Bull. of Univ. of Wis., Eng. Series, Vol. 5, No. 2.

investigations there have been present considerable quantities of impurities which had to be taken into account while in this case these impurities are kept down to a point where the variations observed may be ascribed to the nickel alone.

The hardness tests, as shown, bear out the general qualitative observations made by Burgess and Aston in their work on the same alloys.[3] In the forged condition up to 7 per cent Ni the curve is concave and lies slightly below the value found for pure iron. From 7 per cent to 11 per cent there is a rapid rise to a maximum, dropping off slightly to 19 per cent, and then a rapid drop up to 25 per cent to about the value for pure iron, where it remains practically constant. In the annealed condition the curve starts at about the value for pure iron, rising gradually to 10 per cent, then abruptly to a maximum at 16 per cent, followed by a sharp decline up to 32 per cent, and then

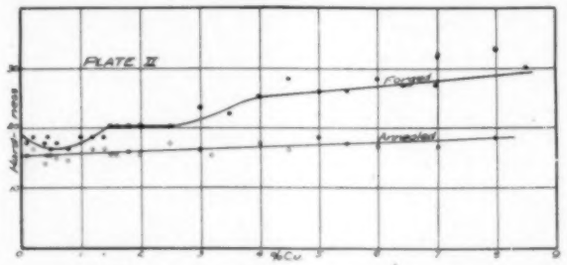


FIG. 2.—HARDNESS OF FORGED AND ANNEALED COPPER ALLOYS.

remains practically constant at a value slightly higher than that of pure iron. As a rule, the variations noted are not large. From 21 per cent to 30 per cent the curve for the annealed samples lies above that for the forged samples. Aside from this, the values found for the annealed samples were about what would naturally be expected as deduced from the curve for the forged samples.

Results on the forged nickel are given in Table I, and on the annealed nickel in Table II, both being plotted together in Plate I.

TABLE I.

Hardness of forged nickel samples.

Per cent Ni.	Number.	Maximum.	Minimum.	Average.
0.25	3	20	15	18
0.5	3	21	17	20
1.0	3	19	15	17
2.0	4	18	17	17½
3.0	5	20	16	18
4.0	5	21	16	18
5.0	5	17	15	16
7.0	5	21	18	19
8.0	5	21	19	20
9.0	6	28	21	26
10.0	5	34	26	32
10.5	3	44	34	39
11.0	8	41	32	38
12.0	5	36	34	35
13.0	6	39	35	37
15.0	3	37	35	36
18.0	3	45	38	37
19.0	2	36	35	35½
20.0	1	..	27	27
21.0	3	40	33	37
25.0	2	20	16	18
36.0	3	20	19	19
45.0	3	20	15	17

Copper-Iron Alloys.

As in the nickel-iron series, quantitative measurements confirm the qualitative observations made on these materials.[4] In the forged condition up to 1½ per cent the hardness curve is concave and lower than that of pure iron. From 1½ to 2½ per cent it proceeds in a straight line, a trifle above that for pure iron, and from 2½ per cent on there is a continued rise. In the annealed condition the hardness increases in a straight line with only a very slow rise throughout the series. In the copper series the variations of the individual readings from the mean were much larger than in the nickel series.

The results from the forged samples are given in Table III and from the annealed samples in Table IV. Both series of results are plotted together in Plate II.

TABLE II.

Hardness of annealed nickel samples.

Per cent Ni.	Number.	Maximum.	Minimum.	Average.
0.25	5	15	14	14½
0.5	5	15	13	14
1.0	4	16	12	14
2.0	4	15	13	14
3.0	5	15	13	14½
4.0	6	15	12	14
5.0	7	17	14	15½
7.0	6	17	15	16
8.0	6	18	15	16
9.0	8	19	15	17
10.0	5	19	17	18
10.5	7	22	18	20
11.0	6	24	18	21
12.0	5	26	23	25
13.0	10	36	26	31½
15.0	3	35	32	34
18.0	6	36	31	34
19.0	5	35	28	31
21.0	2	35	34	34½
22.0	1	..	31	31
25.0	1	..	21	21
26.0	1	..	23	23
35.0	1	..	15	15
36.0	1	..	17	17
45.0	2	19	16	17

TABLE III.

Hardness of forged copper samples.

Per cent Cu.	Number.	Maximum.	Minimum.	Average.
0.1	6	20	17	18
0.2	6	20	16	18
0.4	3	18	17	17
0.45	4	20	17	18
0.5	1	..	16	16
0.6	3	18	16	17
0.8	4	18	15	16
1.0	10	24	16	18
1.2	2	20	16	18
1.4	3	21	17	18
1.5	3	21	19	20
1.6	4	22	18	20
1.8	6	21	19	20
2.0	6	21	17	20
2.5	3	21	19	20
3.0	7	25	21	23
3.5	1	..	22	22
4.0	7	28	22	25
4.5	2	30	26	28
5.0	4	30	23	26
5.5	2	28	25	26
6.0	3	30	25	28
6.45	4	36	25	27
7.0	4	29	24	27
7.5	4	27	24	26
8.0	2	37	29	33
8.5	1	..	30	30

[3] See reference above, 2d, p. 1c.
[4] See reference above, 2d, p. 20.

TABLE IV.

Hardness of annealed copper samples.

Per cent Cu.	Number.	Maximum.	Minimum.	Average.
0.1	8	17	14	15
0.2	8	17	13	16
0.4	3	15	12	13½
0.45	7	16	13	15
0.5	2	17	13	15
0.6	3	15	14	14½
0.8	5	15	11	14
1.0	15	22	12	15
1.2	1	..	16	16
1.4	1	..	16	16
1.5	10	16	13	15
1.6	2	15	15	15
1.8	2	16	15	15½
2.0	13	16	13	15
2.5	4	19	16	17
3.0	14	18	14	16
3.2	1	..	15	15
4.0	11	18	15	17
4.5	3	18	15	16
5.0	8	20	15	18
5.5	3	20	16	17
6.0	6	23	14	16½
6.45	6	21	16	18½
7.0	7	19	14	16½
7.5	9	18	11	15½
8.0	2	19	17	18

General Discussion.

Kuranow and Zemezuzny[5] have attempted to correlate the hardness of alloys with the metallographic condition of the material. Their conclusions are that when two metals form a solid solution the hardness increases and the curve passes through a maximum which generally coincides with the maximum electrical resistance; when the two metals form a mechanical mixture of two components the hardness curve will be a straight line; a definite compound may have a hardness either higher or lower than either of the constituent metals. These conditions are fulfilled, however, in neither the copper nor the nickel series of alloys. In Plates III and IV are plotted the hardness and the electrical resistance for comparison.[6] Curve I shows the

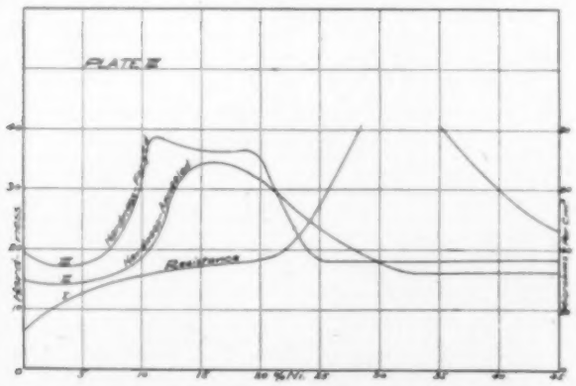


FIG. 3.—HARDNESS AND RESISTANCE OF NICKEL ALLOYS.

resistance in microhms per cubic centimeter, Curve II the hardness of the annealed samples, and Curve III the hardness of the forged samples. In the iron-nickel series a solid solution is formed throughout the series, and a rise in hardness is to be expected. So far the rule holds good, but the maximum hardness is between 10 per cent and 20 per cent, while the maximum in the resistance curve is above 30 per cent.

In the iron-copper series a solid solution is formed up to 3.5 per cent Cu; above that point the two metals are present as a mechanical mixture of two components. According to this a rise of hardness is to be expected up to 3.5 per cent, the curve bearing a maximum at 1.5 per cent, followed by a straight line beyond 3.5 per cent. The hardness, however, in the forged sample lies below that of pure iron, rather than above, the curve bears no definite maximum, and is not a straight line beyond the range of the solid solution. In the annealed samples the straight line curve persists throughout the series.

Portevin[7] refers to the above conditions in his investigation of the electrical resistance of various ternary steels, and after comparing his results with the hardness obtained by Guillet[8] on similar steels, reports a greater variation from this law for the nickel steels than for any of the other materials that he examined.

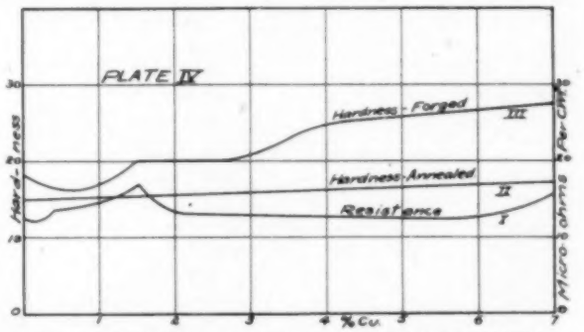


FIG. 4.—HARDNESS AND RESISTANCE OF COPPER ALLOYS.

Conclusions.

From the facts here presented the following conclusions may then be drawn:

1. The addition of nickel to iron gives a maximum range of hardness between 10 per cent and 20 per cent.

2. The addition of copper to iron, up to 8 per cent, gives a gradual, but irregular, increase in hardness.

3. The nickel and copper alloys in the range studied are exceptions to the law of hardness formulated by Kuranow and Zemezuzny.

The results of a similar investigation on the effect of the addition of nickel and copper together to iron are in the course of preparation and will be ready for presentation shortly.

Chemical Engineering Laboratories,
University of Wisconsin.
Madison, Wis.

(5) *Jour. Russ. Phys. Chem. Ges.*, 40, 1067, 1908; *Zeit. Anorg. Chem.*, Oct., 1908; *Jour. Soc. Chem. Ind.*, 93, 1909.

(6) Values for electrical resistance are those measured by Burgess and Aston for these same samples; see reference 2b.

[7] Rev. de Metall., 6, 1304, 1909.

[8] Les Aciers Speciaux, Dunod, Paris, 1904.

The Northern Chemical Engineering Laboratories is the designation of an enterprise recently started at Madison, Wis., the purpose of which is to carry on experimental and investigation work along lines of chemical engineering and applied electrochemistry. This is under the direction of C. F. Burgess, professor of chemical engineering, University of Wisconsin. Associated with him are Carl Hambuechen, formerly chemical engineer for the Aluminium Company of America; James Aston, formerly owner and manager of a metal foundry business in Milwaukee and later instructor in chemical engineering, University of Wisconsin, and O. E. Ruhoff, recently research chemist for the Mineral Point Zinc Company. It is proposed to continue some of the more promising lines of investigation which have been begun in some of the college laboratories, and the lines of work to which especial attention is being given include the corrosion of metals, metallography, and the investigation of the production and properties of alloys of iron; dry cells, chemical products and various types of machinery and apparatus used in chemical engineering work.

Elementary Principles of the Designing and Proportioning of Electrical Furnaces.

BY CARL HERING.

The fundamental principle of an electric furnace is so extremely simple, namely, the generation of heat by a current, that one is apt to think that the design of a furnace is an equally simple matter. It is true that heat is a low form of energy to which all other forms naturally degenerate, hence the conversion of the higher forms of energy, like chemical energy (of coal, for instance) or the highly serviceable form called electrical energy, into the degenerated form called heat, is decidedly simpler and generally far more efficient than the reverse. Hence an electrical furnace would naturally be a simpler device than one which performs the reverse operation. But simple as it may be in its fundamental principle, there are many details of design and proportions which make the difference between a good and a bad furnace, and there are systematic and direct methods of design as distinguished from the haphazard, cut-and-try, or roundabout methods which are often resorted to when the fundamental principle is such a simple one. When the necessary constants are known there is no reason why furnaces should not be proportioned rationally and with care, so as to get the best results possible, although the accuracy will never reach that of dynamo design, because there exist no heat insulators like those used to insulate electricity.

The purpose of the present article is to summarize briefly the principles involved in furnace design as far as the proportions are concerned, and to describe the proper sequence in making the calculations, so as to get the best results in the most direct and rational way. Also to point out the ideal conditions which we should strive for even if we can only approach but never reach them.

While a very definite class distinction is often made between arc and resistance furnaces, which distinction is important on account of details, the difference is in fact not so great as might at first appear. The arc is really a gaseous resistor, and the heat is generated by the current passing through these gaseous resistances, which appear to have a very high specific resistance, thereby enabling a very large amount of heat to be generated in a very small space, hence high temperatures; and it is this that gives it its desirable qualities for furnaces. In this feature, therefore, it is merely a resistance furnace with a very small resistor.

There is a characteristic difference, however, in the fact that an arc produces a true counter electromotive force besides the drop of voltage due to resistance; in this it is somewhat like self-induction coils, motors, electrolytic baths, etc. The energy represented by the product of the current and this fractional part of the total voltage is not primarily set free as heat and must not necessarily be included in determining the heat evolved, although often it may finally degenerate into heat even in the furnace itself, in which case it may become available.

This counter-e.m.f. in the arc always represents some other form of energy into which this part of the electrical energy is transformed. In most, if not all, simple arcs it represents that energy which is necessary to volatilize the carbon or other electrode material—that is, to change its physical state. If this vapor passes off as such without condensation or combustion (as, for instance, in a zinc or arsenic furnace) it is lost as far as its heat equivalent is concerned. If it combusts in the furnace its combustion heat is added and may become available. This counter-e.m.f. factor should, therefore, be taken into consideration if large enough; in other respects most of the thermal calculations for arc and resistance furnaces are alike.

The first step in the design of a furnace is to decide upon the output and, of course, the rate per hour, for neither the size of a furnace nor the power input will depend on the former alone; the latter must be specified. A small furnace with a relatively high power input may produce as much product as a large one with a relatively low power input. The choice must be based on the particular process; some processes may be analogous to the hatching of an egg; if hurried, the result will not be the one desired. In general, however, having decided on the desired product per day, the advantages are decidedly in favor of the smallest possible furnace that will yield this product. In other words, the furnace should be relatively small so that it will be forced as much as other considerations permit. The chief reasons are that a small furnace is not only cheaper, but it enables the heat losses through the walls to be reduced.[1]

This should not be confused with another fact, namely, that per cubic inch of internal space the heat loss through the walls is much less for a large than for a small furnace.[2] This means that (other conditions being the same) it is more economical, per pound of material, to produce a large output than a small one, and that it is much better to produce the whole output in one large furnace than in several smaller ones.

The power plant is, of course, smaller the more steadily the furnace is run during the 24 hours. The energy required to heat up a furnace to the stable temperature will probably be found to be relatively very great, and as this is all non-productive it is, of course, much better to use a continuous rather than an intermittent process, or with the latter, to lose as little time as possible between the charges. With correctly proportioned electrodes the loss through the electrodes when the current is stopped, is almost exactly half of that when normal current flows; when not correctly proportioned this stand-by loss may be much greater than one-half.

The next step is to calculate the theoretical amount of energy required for the chemical and physical changes of the specified amount of the product, as that will represent the least possible energy. This, of course, is independent of the furnace design and includes the heat necessary to raise the raw material as it enters to the temperature at which the final products leave, therefore involving the specific heat, generally the latent heat of liquefaction, and sometimes also that of vaporization; to this must be added or subtracted (more generally the former) the energy of chemical combination, if any; that is, the energy represented by the chemical changes which may take place in the furnace product. If this change absorbs energy, the process is an endothermic one and the energy must then be added to the other quantities to give the total which the furnace must supply. The resulting product then is of a higher order, on a thermochemical scale—that is, the product stores up a part of the furnace energy in itself while in the furnace, and is then generally in a condition in which it can again give off this energy later. The production of the metals, iron, aluminium, sodium, etc., from their ores, the production of calcium carbide, the production of steel from cast iron, etc., are all processes of this nature. In fact, it seems that most electric furnace products are of this nature, and sometimes this chemical energy is relatively very large and therefore very important in the calculation.

If, on the other hand, the chemical change in the furnace gives off energy, then that much energy may be subtracted; it is then called an exothermic process. This is less frequently the case, as such processes generally do not require an electrical furnace, because many of them will go on by themselves after starting. If the process is a composite one in which several chemical changes take place, some of which may be thermally positive and some negative, they should, of course, all be included. In simple smelting there is no such chemical change, except that between the metal and the slag, which may be very small.

Besides the energy for these primary chemical changes there may also be some secondary or subsequent thermochemical reactions, such as the heat of combustion of carbon vapor, which may be worth considering.

[1] "Heat Conductances Through Walls of Furnaces." Hering. Trans. Amer. Electrochem. Soc. Vol. 13, fig. 5, p. 225.
[2] Ibid. fig. 6, p. 225.

Having thus determined the least possible quantity of energy required for both the physical and the chemical changes, estimate the first approximation to the electrical input in watts by allowing for the loss of heat by conduction through the walls, and that necessarily lost in the electrodes. The former depends, of course, greatly on the construction of the furnace, its size, etc. Efficiencies of 80 per cent have been claimed, but 50 to 60 would probably be nearer the truth, unless care has been taken in the design.

The next step is to decide what the current and voltage are to be, as so far only their product is known. This also depends greatly on the construction of the furnace, and must be governed by it. Thermically, of course, it makes no difference. The loss in the electrodes increases with the current and is independent of the voltage, hence, if this loss, which may be made very small by proper design, is of any importance, it is best to make the voltage as high as possible, thereby reducing the current. But as this difference is small in percentage, there will probably be other practical considerations which will outweigh it, and, therefore, govern the choice.

In an arc furnace the arc should be made as long as other considerations permit, and two arcs in series would be better than one, as this halves the current. In a resistance furnace the reduction of the current at the expense of the voltage makes long and slender resistors, which are as a rule very objectionable and impracticable. The calculation of the length and cross-section of a resistor of a given material, to set free a certain amount of heat, is too simple to need description here.

The next step is to determine the size of the interior of the furnace. The only rule is to make it as small as practicable to hold the material. Waste space in the interior of a furnace diminishes the efficiency greatly.[3] By properly proportioning the electrodes and the walls so that they will not chill the furnace product, the whole of the interior of a furnace becomes available, hence it may be made much smaller than when the material is being chilled, and it seems likely that in most existing furnaces there is such a chilling action, due to the electrodes, because our former rules for proportioning electrodes tend to make them much larger than necessary, or too short, both of which cause not only a useless loss of heat by conduction, but reduce the available space in the furnace on account of the space taken up by the chilled material.

The correct proportioning of electrodes assumes that the hot end of the electrode can withstand the temperature of the furnace. In some very high temperature furnaces this may not be possible either for the electrodes or for the walls, and in such cases the only way seems to be to allow some of the product next to the walls to remain inactive by conducting off the heat from it as fast as it reaches the material, as is the case, for instance, with the water-cooled tuyers of blast furnaces around which some slag chills and remains solid although in contact with molten slag. In such a case this material must be considered as a part of the walls, like in the carborundum furnace, and not a true, available part of the interior of the furnace proper. Or when such material is at the end of the electrode and is conducting, it must in the same way be considered as part of the electrode which ends (when correctly proportioned) at that place where there is no flow of heat from the useful product out through the electrode—that is, where the temperature equilibrium between the electrode and the furnace is reached.

Having thus established the actual interior size of the furnace, its shape and wall thickness must then be decided upon. As to the shape, the only general rule is that a sphere is theoretically the best, as far as the heat loss through the walls is concerned. Hence, if that loss is of importance the nearer it approaches to a sphere the better; hence, a cubical shape is better than a long rectangle or long cylinder. But in this, as in many other features, practical considerations will often outweigh the gains due to the theoretically best conditions; but it is well to know what the latter are so that they can at least be approached as closely as practical conditions permit.

The thickness of the walls and the loss of heat through them is then determined. The paper referred to above gives further information about this as well as about the question of the best shape. In these calculations it should be remembered that the loss of heat through the walls depends on the *ratio* of the cross-section (perpendicular to the flow of heat) and the thickness, as distinguished from the actual dimensions—that is, the latter may be greatly different in two cases and yet the loss be the same if that ratio is the same.

Having determined the wall thickness, it generally establishes the necessary length of the essential part of the electrodes—that is, the part which must exist to get the current into and out of the furnace. It is this part in which the current may be made to heat the electrode tip, to just the furnace temperature so as to check the outflow or loss of any furnace heat through it, under which conditions the total loss in or through electrodes will be either exactly a minimum or very nearly so. This establishes the cross-section and the loss in the two electrodes. This part of the calculation has been described so fully in other papers by the writer[4] that it is not necessary to repeat it here.

Whatever additional length the electrodes may have must be determined from other considerations, remembering that the electrical resistance heat in the external parts is always a dead loss. Hence, if feeding is required it is best, if possible, to place the terminal close to the wall and feed the electrode throught it, or to make the feeding length as short as is consistent with the labor cost of moving the electrode through its terminal. In the latter case it is preferable to increase the cross-section and therefore also the essential length, as much as practicable, remembering, however, that the essential part must be well embedded in the walls or else the losses will increase and perhaps very greatly.

This completes the first trial calculation. The losses which were assumed at the start have now been calculated more accurately (provided the necessary constants exist) and if the assumed ones are found to be materially different, the proportions may be corrected accordingly, if necessary.

If, now, in the operation of the completed furnace it is found that the necessary temperature is not reached (after giving ample time for the stable state to be reached, especially at first when the brick work may be damp), it means that either the feeding of the material is too rapid or that the losses are too great, or both; in either case it shows that the constants were poor or that an error has been made in the calculations. The temperature will always rise until the heat absorbed or lost is equal to that produced. If, on the other hand, the temperature rises too high, then simply feed in more material, as the furnace then has a greater capacity than that which was assumed.

If the furnace product is found to be chilled around the ends of the electrodes, they are too large in section, or too short; increasing the current would remedy this. If they burn out the walls, because their temperature is higher than that of the furnace, they are too small in section or too long; diminishing the current will correct this.

It is self-evident that the losses in the walls and in the electrodes become greater for higher temperatures; hence, it is a disadvantage to develop a higher temperature than is necessary for the desired chemical reaction or physical change to take place. Therefore, as far as this is concerned, a resistance furnace would generally be better than an arc. The temperature of an arc is believed to be that of the vaporization point of the material of the electrode, hence generally very high and far higher than necessary. On the other hand, the rapidity of the

[3] See reference cited above, p. 224, near bottom, referring to fig. 5.

[4] In the most condensed form in an article entitled "Simple Rules for Proportioning the Electrodes for Minimum Loss," *Electrical World*, June 16, 1910, p. 1598, in which references to other articles are also given.

transference of heat to the product increases when the temperature of the source of heat is much higher. The ideal method would be to generate only that temperature which is required for the chemical reaction, or only slightly more, and then to transfer the heat by rapid circulation; hence, heating from the bottom would be far preferably than from the top, as gravity then assists in this transference of the heat. It seems to be a very slow and expensive method to rely entirely on the differences of temperature for the necessary transference of heat.

This rapidity of transference of the heat may often be a very important factor in determining the rapidity of the treatment of the material, on which in turn the size of the furnace depends; hence, it deserves careful attention. The inavoidable heat losses through the walls and in the electrodes are, of course, continuous, hence, if a pound of the product goes through the furnace twice as fast, for instance, these losses will be only half as great per pound of product.

It is also better, and perhaps far better, wherever possible, to transfer heat from a good conductor (metal) to a bad one (slag) instead of the reverse, as the extraction of the heat from a thin layer of a good conductor in contact with a poor one tends to leave it a still better conductor; while the reverse tends to make most poor conductors still worse approaching an insulator; hence, in the latter case the rapidity of flow will tend to be checked, while in the former it will tend to be augmented.

FIG. 1.—TRAIN LOADED WITH GAS-ENGINE BLOWING UNIT.

The limit of the temperature in a resistor which remains solid, like carbon, seems to be its vaporization temperature, like in the arc, although in practice the rapid oxidation, and deterioration due to other causes, its fragility at high temperature, etc., may determine a far lower limit. The limit of temperature in a liquid resistor is for most materials determined by the "pinch phenomenon" discovered by the writer some years ago,[8] which causes the column to part when a certain current is reached, after which the product freezes on open circuit.

In conclusion the writer desires to repeat that, as in most engineering designs, practical considerations will often determine some of the questions for the designer, or will limit him in his efforts to approach the theoretical ideals; that is no reason, however, why we should not try to find out what these ideal conditions are, and to keep them in mind so as to approach them as closely as practical consideration will permit, even though at times we may not be able to get very close to them. And unless we know what the theoretically best or limiting conditions are we may be attempting something which is impossible or commercially impracticable.

[8] "A Practical Limitation of Resistance Furnaces; The 'Pinch' Phenomenon." Trans. Am. Electrochem. Soc. Vol. 11, 1907, p. 329. "The Working Limit in Electrical Furnaces Due to the Pinch Phenomenon." ibid. Vol. 15, 1909, p. 255.

Two-stage crushing is being adopted in South Africa, using heavy stamps of 1800 to 2000 pounds for the first stage and tube mills for the second. The stamps crush to 4 or 8-mesh, and have a capacity of about 20 tons per day. The tube mill grinds this product to the degree of fineness considered necessary to free the gold.

Gas Power for Canadian Steel Plant.

During the past few months Allis-Chalmers Company has shipped four gas-engine generating sets and two gas-engine blowing units to the Lake Superior Iron & Steel Corporation at Sault Ste. Marie, Canada, and has two more of the blowing units nearing completion. Each one of these units required twelve cars for its transportation and its weight approximated 1,000,000 lb. Fig. 1 shows a train loaded with a single gas-engine blowing unit.

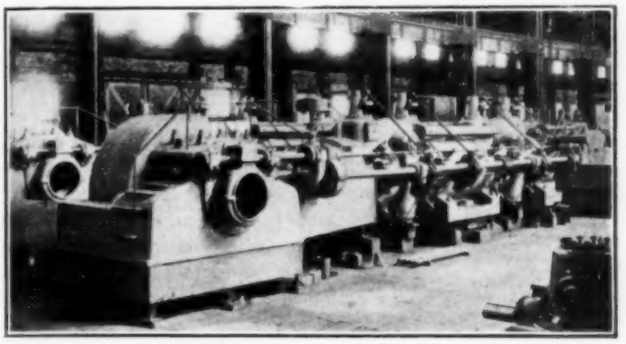


FIG. 2.—34-48 GAS ENGINE.

These gas-engine units are being installed in a large extension of the Lake Superior Iron & Steel Corporation's plant at Sault Ste. Marie, Ontario. The original rail mill will have a largely increased capacity and new plate and merchant mills will be built as well as coke ovens. For furnishing power for the mills and blowing the furnaces the company purchased the above units from Allis-Chalmers Company. They will be supplied with gas from the blast furnaces.

The gas engines (Fig. 2) are all alike, being of the twin-tandem, four-cycle type with cylinders 34 inches x 48 inches. They conform to the company's standard in all respects. Four of these engines are direct-connected to 1765-kva, 25-cycle, three-phase, 2300-volt alternators running at 107 r.p.m. These sets will supply power for driving the motors about the mill.

The other four units are to be connected to the new Slick blowing tubs (Fig. 3) manufactured by Allis-Chalmers Company. These tubs are 64 inches x 48 inches and are arranged to operate duplex on the opposite side of the main shaft from the engine. Each unit has a capacity of 25,000 cu. ft. per minute at 72 r.p.m., but can be speeded up to 85 r.p.m.

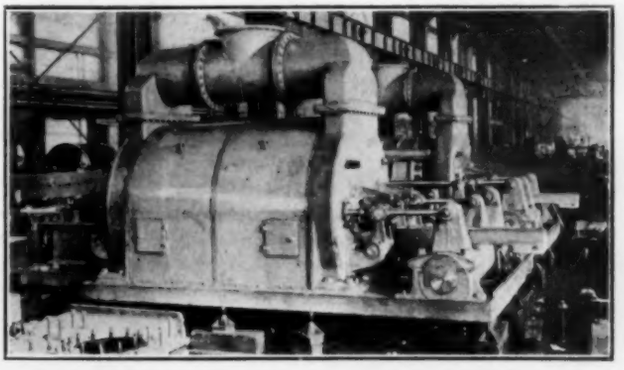


FIG. 3—SLICK BLOWING TUBS.

When the new works are completed this will be the largest steel-producing plant in Canada, and naturally the most modern. Mr. Alfred Ernst has been consulting engineer on the work and much of its success will be due to his efforts.

The Bureau of Mines.

The act establishing a Bureau of Mines in the Department of the Interior, approved May 16, 1910, became effective July 1. As originally approved, the law contemplated the transfer of the entire Technologic Branch of the United States Geological Survey, the mine accident investigations, fuel investigations, structural materials investigations, the entire personnel, property and equipment to the Bureau of Mines, but the sundry civil appropriation act, approved June 25, amended the law to such an extent that the structural materials investigations, including the personnel and equipment for these investigations, went to the Bureau of Standards, Department of Commerce and Labor.

Carrying out the spirit and intent of the law so amended, the Secretary of the Interior has transferred to the Bureau of Mines the investigation of mine accidents and fuels, together with the personnel and equipment of these investigations, and has transferred to the Bureau of Standards the structural materials investigations and the employees of the Technologic Branch of the Survey engaged in these investigations. The fully equipped testing station at Pittsburgh also goes to the Bureau of Mines.

The Bureau of Mines, therefore, includes the mine accidents and fuel investigations, for which an appropriation of $410,000 was made by Congress. The total appropriations for the bureau, including salaries, rent and expenses of removal amount to $502,200.

The work of the Bureau of Mines for the first year will be a continuation and expansion of the work carried on by the Technologic Branch of the Geological Survey. The law in itself provides for a variety of other problems that properly belong to the Bureau of Mines and which should eventually be undertaken, such as methods of mining and metallurgical processes, but these activities will be deferred for the most part until Congress gives additional authorization in the shape of adequate appropriations. The spirit of the debates in Congress, both on the Bureau of Mines legislation and on the appropriation items, emphasized the desire to regard the mine accidents investigations as urgent and this will be the feature of the work.

In all, $310,000 was appropriated for mine accident investigations. Of this sum under the general plans approved by the Secretary of the Interior, $120,000 is to be spent on the rescue stations, $36,000 for housing nine stations, $34,000 for equipping eight new stations, and $10,000 for additional equipment for five existing stations. The allotment for the investigation of explosives is $40,000; for electricity in mining, $14,000; appliances for preventing mine accidents, $8,000; examination and codification of mining laws, $5,000, and other technologic investigations, ore treatment, etc., $10,000.

For the analyzing and testing of the coals, lignites, ores and other mineral fuel substances belonging to or for the use of the United States, $100,000 was appropriated. Of this amount, $35,000 will be spent in the chemical and physical investigation of fuels; $25,000 in the inspection of Government fuel purchase; $22,000 in fuel efficiency investigations; $5,000 in lignite and peat investigations, and $4,000 in briquetting investigations.

The mine accident investigations, which have been transferred from the Geological Survey to the Bureau of Mines, were first authorized in the legislative appropriation act of May 22, 1908, carrying for this purpose an appropriation of $150,000. This was followed by a similar appropriation carried in the act for the sundry civil expenses of the Government for 1910. A mine experiment station was established in Pittsburgh during 1908, at which, since that time, investigations of explosives, coal gas, dust, electricity and other possible causes of mine explosions have been continually under way. The mining engineering field forces of the Geological Survey has already made decided progress in the study of underground mining conditions and methods. Practically all of the coal mines in which mine explosions have occurred during the last two years have been carefully examined, the gases, coke and dust have been analyzed at the laboratory at Pittsburgh and every effort has been made to determine the explosibility of various mixtures of gas and air in the presence of shots of different types of explosives.

Considerable progress has also been made in the investigation of explosives used in coal mining, and the conditions under which they may be used with least risk. Manufacturers have submitted many explosives for test at the station and a considerable portion of them passed and have been classified among the permissible explosives. The investigations and educational work in connection with the use of artificial breathing and other types of mine rescue equipment, the so-called oxygen helmets, have not only been useful in developing a more satisfactory use of such equipment in the examination of mine explosions, but also better methods for using this equipment in mine rescue work.

The fuel investigations under the Geological Survey, and which are transferred to the Bureau of Mines, have already resulted in a better realization throughout the country as to the value of fuels. One result of this work is that nearly all of the fuel now purchased by the Federal Government is bought on specifications and subject to test by the fuel division, or purchased after examination made of the coal supplied by the mines from which coal is delivered to the Government.

The publications of the Survey relating to mine and fuel investigations, those prepared by the Technologic Branch, will in the future be distributed by the Bureau of Mines. The publications relating to structural materials will continue to be distributed by the Geological Survey. The last of the bulletins of the Technologic Branch to be published by the Survey will be issued from the Government Printing Office about Aug. 1. This bulletin relates to the explosibility of coal dust and was prepared by Mr. G. S. Rice, with chapters by Messrs. J. C. Frazer, Axel Larsen, Frank Haas and Carl Scholz.

The first of the Bureau of Mines bulletins, "The Voltatile Matter of Coal," by Messrs. H. C. Porter and F. K. Ovitz, will be published in the next few months. Then will follow "Coal Analyses," by Messrs. N. W. Lord and J. S. Burrows; "Final Data Regarding Steam Tests," by Mr. L. P. Breckenridge; "North Dakota Lignite as a Boiler Fuel," by Messrs. D. T. Randall and Henry Kreisinger; "Producer-Gas Tests in 1905-1907," by Mr. R. H. Fernald; "The Coke Industry as Related to the Foundry," by Mr. Richard Moldenke; "Coals for Illuminating Gas," by Messrs. A. H. White and Perry Barker; and "Petroleum for Combustion Under Steam Boilers," by Mr. I. C. Allen.

These publications, when issued, can be obtained by addressing the Director of the Bureau of Mines, Washington, D. C.

Effects of Ultraviolet Rays on Gases.

M. M. Berthelot and Gaudechon have investigated the effect of ultraviolet rays from a quartz mercury-vapor lamp on acetylene, cyanogen, ethylene and oxygen, and in every case found a decrease in volume consequent on polymerization, but did not observe any other chemical change. Acetylene is transformed into a yellow solid, and the change takes place more readily under reduced pressure or when an inert gas is admixed; ethylene yields a liquid polymer having an odor resembling that of rancid fat; cyanogen becomes paracyanogen and oxygen is to a limited extent converted into ozone.

The Fifth International Congress for Mining, Metallurgy, Applied Mechanics and Practical Geology.

By Jos. W. RICHARDS

Düsseldorf, the vigorous young city which "watches on the Rhine" as the representative city of the great Westphalia coal and iron industry, was selected by the fourth congress, at Liège, for the congress of 1910. How the Westphalians, technically the most progressive people in Germany, rose to the occasion and surpassed all expectations, we will try to tell.

The congress was a model of careful forethought, system and minute attention to detail, to a degree which astonished even the Germans themselves from other provinces. And besides time and thought, money was provided in abundance for every purpose; it was said that the city of Düsseldorf itself contributed $50,000 to entertain the visitors. Fifteen hundred members were in attendance, and yet the extraordinary ability of the organizers and conductors of the congress assured everyone a comfortable and profitable time.

HISTORY OF THESE CONGRESSES.

The first International Congress of Mining and Metallurgy was organized by the Société de l'Industrie Minerale of St. Etienne, France, in connection with the International Exposition of Paris in 1878. Only about 150 persons took part in this congress, most of them from France.

The second congress was invited by the French Government to meet in Paris, in connection with the exposition of 1889. Over 600 accepted the invitation, among them many from outside France.

The third congress, called together in connection with the Paris exposition of 1900, was attended by representatives from almost all civilized countries.

The fourth congress was held at Liège, in 1905, in connection with the International Exhibition. Here it was determined to unite in one congress the fourth Congress for Mining and Metallurgy with the third Congress for Applied Mechanics (the two previous ones had been held in Paris in 1889 and 1900) and the first Congress for Practical Geology. It was thus that the congress attained its present scope.

The fifth (and present) congress was held in Düsseldorf on June 20 to 25, under the auspices of the *Verein für die bergbaulichen Interessen im Oberbergamtsbezirk Dortmund* and the *Verein deutscher Eisenhüttenleute*. At their invitation 1910 members enrolled, of whom 654 were from outside Germany, and about 1500 were in actual attendance. The membership of the four sections were: Mining, 1178; Metallurgy, 1177; Applied Mechanics, 972; Practical Geology, 810. This congress was the first to meet unconnected with an international exhibition, but at the close many of those present participated in a three-day excursion to the Brussels exposition.

The sixth congress will be held in London in 1915.

SOCIAL FEATURES OF THE DUSSELDORF CONGRESS.

On the evening of June 19 an informal gathering of members and friends was held in the Tonhalle; the ladies were given a separate reception in the Art Gallery.

On the evening of June 20 the city of Düsseldorf gave a reception to the congress in the Tonhalle. This was accompanied by music and refreshments served in the fine gardens, and was a very enjoyable function. The afternoon of this day the ladies were shown the sights of the city.

On the 21st the ladies were taken on an excursion to Zons in the morning and in the evening a banquet was given by the city in the two great halls of the Tonhalle, at which over 1000 were present. At the close a number of fine after-dinner speeches were made in German, French and English, and the festivities closed with a wonderfully clever bit of theatrical staging wherein was shown the interior of the International Steel Works, with Herr Gärtner, of the Düsseldorf Stadttheater, as Obermeister, and Frau Paschke as the Genius of Technique. The pouring of the steel and casting of the ingots was finely done, and the conversations of the workmen and declamations of the chief actors were very well performed.

On the 22d the ladies were taken on an excursion into the hill country around Solingen in the morning, while the evening was given over to a steamer trip on the Rhine, followed by illumination of the banks and a splendid exhibition of fireworks. On this day Count Zeppelin traveled from Switzerland to Düsseldorf in his great airship, making the 450 miles in nine hours. Opportunity was given to see the splendid balloon, and in the evening Count Zeppelin accompanied the members on the steamer, where he was most enthusiastically received and conversed with many of the "congressistes."

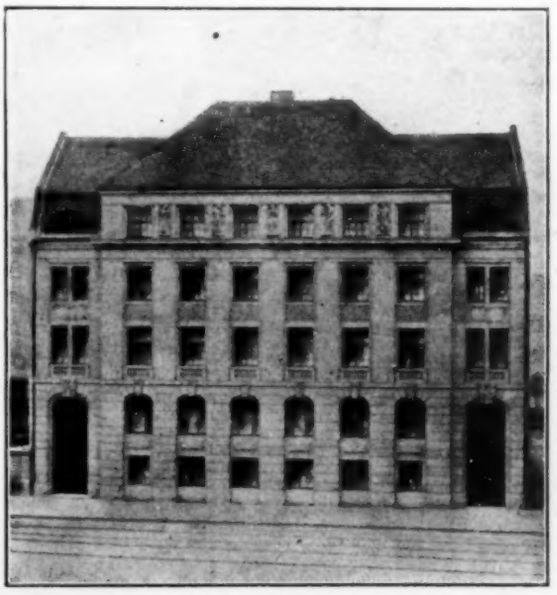


FIG. 1.—OFFICE BUILDING OF THE SOCIETY OF GERMAN IRONMASTERS.

On the 23d the ladies were shown the sights of Essen in the morning, and in the evening, after the closing sitting of the congress, there was given a splendid "Gartenfest" in the City Hall of Essen, with dancing and illumination of the park.

The 24th, 25th and 26th were devoted to the excursion to Brussels, made from Düsseldorf in two special trains, and which included a reception in the German Exhibition Building by the German commissioner, on the 24th; a banquet, followed by fireworks, given by the exposition commissioners, on the 25th, and a reception by the city of Brussels, in the magnificent Town Hall, on the 26th.

Fig. 1 shows the new office building of the Society of German Ironmasters (Verein Deutscher Eisenhüttenleute); Fig. 2 the office building of the German Steel Trust in Düsseldorf; Fig. 3 the crucible steel department of the Krupp Steel Works in Essen.

VISITS AND EXCURSIONS.

A finely printed book of 100 pages, printed in both German, French and English, was provided for describing the 40 different excursions and visits planned. Of these, 35 were visits to industrial plants, and five were geological excursions. Of the former, there were open to visiting: 1 river improvement

FIG. 2.—OFFICE BUILDING OF GERMAN STEEL TRUST.

plan, 2 harbor and dock-yards, 1 electric power station, 1 technical high school (Aachen), 8 machine works, 20 collieries, 12 coking plants (all by-product), and 14 iron and steel works. Each excursion was accompanied by a guide and was run on a time schedule. The members taking the excursions were not even asked to pay for the railway tickets, all being provided for.

SCIENTIFIC WORK OF THE CONGRESS.

In the four sections there were altogether 120 communications, 35 on mining, 44 on metallurgy, 23 practical metallurgy, 21 theoretical, 17 on applied mechanics and 24 on practical geology. In the writer's opinion, the congress would gain in interest and usefulness if its field were not so broad; if, for instance, the section "Practical Geology" were turned over to the International Geological Congress, which meets every four years, this year in Stockholm; the section on "Applied Mechanics" should form the nucleus of an international mechanical congress; and the sections which remain—Mining and Metallurgy—representing the original Congresses I, II and III, would be quite sufficient to constitute an international congress of themselves. The dragging in of the other sections divides interest in the congress and makes it ungainly in size.

Geological Papers.

Of these papers, the only ones which would deeply interest metallurgists and chemists are:

The Iron Supplies of the World, by Dr. Beyschlag, of Berlin. This was not in print. Metallurgists interested are referred to the great compilation on this subject which is now being printed in two volumes in Stockholm by the International Geological Congress.

The Utilization of Water Powers, by Holz of Aachen. This was also not in print in time for distribution at the congress.

The Mineral Resources of Canada, by H. Mortimer-Lamb, of Montreal. A condensation of the reports on mineral resources, such as are issued by the Canadian Government, showing an increase in mineral production from $10,000,000 in 1886 to $90,000,000 in 1909; the increase *per capita* product being from $2.23 in 1886 to $12 in 1909. The water-powers of Canada are given as follows:

	Possible horsepower.	Developed horsepower.
Nova Scotia	54,300	13,300
New Brunswick	150,000	....
Quebec	10,000,000	700,000
Ontario	6,190,000	350,000
Manitoba	495,000	18,000
Northwest Territories	600,000	....
Saskatchewan	500,000	....
Alberta	1,144,000	1,330
British Columbia	2,065,500	73,100
Total	21,198,800	1,155,760 (= 5.5%)

The Canadian Government pays the following bonuses on pig iron and steel manufactured electrically in Canada:

	Pig iron.	Steel.
During 1910	$2.10 per ton	$1.65 per ton
During 1911	1.70 per ton	1.05 per ton
During 1912	0.90 per ton	0.65 per ton

It is interesting to note that if the steel is made electrically from electrically produced Canadian pig iron, both bounties can be collected.

Mining Papers.

Of the 35 papers on mining the most important were probably those of Engineer Zaeringer on the latest developments in sinking shafts by the freezing method, and Bergassessor Viebig's excellent treatise—one might almost call it—on the use of reinforced concrete in mine construction. For us, the following only was of metallurgical interest:

The Use of Low-Grade Fuels, by Bergassessor Dobbelstein, of Essen, considered the question of burning coal of the following kinds:

FIG. 3.—CRUCIBLE STEEL DEPARTMENT OF KRUPP STEEL WORKS, IN ESSEN.

1. Coarse coal, with up to 50 per cent ash and up to 20 per cent water,
   (a) with high volatile matter,
   (b) with small volatile matter.
2. Fine coal, with up to about 40 per cent ash and up to 30 per cent water.

(a) with high volatile matter,
(b) with small volatile matter.

3. Coke breeze (fine coke), with up to 30 per cent ash and up to 20 per cent water.

The Essen Boiler Inspection Society with the Dortmund Mining Society contributed a fund to investigate the possible use of these materials. The projected gas-producer tests have not yet been made, but the boiler tests have been completed with valuable results. A simple rule for comparing the values of different fuels was arrived at, viz., the values in pfennigs (1 pfennig = 0.25 cent) per ton equals the calorific power divided by the per cent of ash plus half the per cent of water in the coal. The quotients express with satisfactory accuracy the relative industrial values of the respective fuels.

If briquetted, they should be pressed into small cylinders 2 in. in diameter by 2 in. long, which quickly become ignited and stay together, while larger briquettes warm up so slowly that the pitch escapes and they fall to pieces. Tar-pitch residues should be used, which can be bought in Germany for about 9 marks ($2.25) per ton. Coke breeze was likewise burned on a perforated grate with blast and by the Praesto firing.

The Praesto firing showed up cheapest and most effective, giving 1 ton of steam for 0.98 mark (24.5 cents), and evaporating nearly 23 kg of water per hour per square meter (4½ lb. per square foot) of heating surface. Experiments with the poor, coarse fuels on ordinary grates showed that in many cases it was more advantageous, regarding cost per ton of steam produced, to use the poor fuel than ordinary good fuel.

Metallurgical Papers.

Electricity in Metallurgical Work was discussed by K. MALEYKA, of Berlin, in a very general manner. He gave examples of the different systems of electrical energy generation and distribution and the different types of motors suitable for various metallurgical operations, and gave particulars of the energy requirements for separate purposes in blast-furnace, steel and rolling-mill service. He concludes that it is usually necessary, in the designing of electric driving machinery, to consider each case by itself.

The Present Status of Ore Dressing, by Bergassessor P. BODENSTEIN, of Kalk, near Cologne, starts out by saying that hand-sorting will probably remain always the first necessary treatment, so as to remove coarse ore and gangue. High-speed rolls, wet ball-mills and wet pendulum grinders are all commended, while wet edge-mills and stamp mills, although releasing the ore well, are expensive to install and to work because of their small output.

Electrostatic separation is only effective on completely dry, fine-grained and at the same time dustless ore; its application is limited, and it is used only where other methods are ineffective. Spathic iron ore is being successfully roasted in revolving tube furnaces, and more recently it is being treated in "roasting converters" (Bessemer roasting, or blast roasting, or blast sintering, however it may be called), constructed by the Humboldt Machinery Company.

Wire-Rope and Electric Suspended Railways are described by Oberingenieur G. VON HANFFSTENGEL, of Leipzig. He particularly recommends for use in metallurgical works the "electric rope-way system," which is a combination of the wire-rope railway with the electric suspended railway on the Bleichert system. This combination possesses the advantages of the rope railway in surmounting steep grades with the adaptability of the electric service for turning corners, and is recommended for charging cupolas and blast furnaces. Since elevations of any height can be surmounted, the system can be used for piling material into storage heaps, which it performs at a cost of about 2 cents per ton.

Wire-Rope Railways for Heavy Service were discussed by Direktor ELLINGEN, of Cologne. Lines up to 37 km (22 miles) length are in operation; the four-wheel bogie has mostly displaced the two-wheel for heavy transport. The firm of Pohlig has built lines which carry 250 loads per hour, of 2 tons each, or a total of 500 tons per hour, at a cost of 1.5 to 2 pfennigs (0.4 to 0.5 cent) per ton-kilometer (0.6 to 0.8 cent per ton-mile). The author believes that by using four-wheel supports the cost may be brought down to half the above-given maximum, i.e., to 1 pfennig per kilometer.

Electrical Lifting Appliances in Metal Works were extensively described by Engineer F. HEYM, of Wetter (Ruhr). He particularly dwelt on the increased speed and increased output made possible by the use of fast-moving electrical machinery.

The Changes in Fire Bricks During Use, by Dr. BLASBERG, of Dahlhausen (Ruhr), deals with the changes due to heat, the action of slags, the action of gases, deposition of carbon, ash of the coal, etc. The author made numerous analyses of the slagged or glazed part of the bricks, and compared them with analyses of the unaltered kernel of the brick.

The work covered a large range and gives much valuable information. Of the 86 complete analyses recorded in the paper, we cite the following representative ones, and repeat some of the author's conclusions:

	SiO_2.	Al_2O_3.	Fe_2O_3.	CaO.	MgO.	$(K_2Na)_2O$.	
Cupola brick:							
Kernel..	89.66	7.79	0.69	0.33	0.17	1.20	(MnO.)
Slagged.	72.59	7.24	11.24	0.28	1.47	4.61	2.57
Blast-furnace brick:							
Kernel..	59.79	25.50	3.45	0.56	0.36	10.34	...
Slagged.	53.50	27.17	5.34	0.88	0.52	12.59	...
Copper-furnace brick:							
Kernel..	64.13	29.61	1.89	0.80	0.72	2.85	(CuO.)
Outside.	63.95	29.67	1.93	1.10	0.75	2.50	0.10
Slagged.	52.10	19.81	4.39	0.52	0.47	3.90	21.66
Blast-furnace brick:							
Kernel..	69.54	25.34	1.20	0.50	0.54	2.68	...
Crust...	54.18	25.45	1.25	0.90	0.57	17.65	...
Coke-oven brick:							
Kernel..	84.99	11.28	1.02	0.34	0.31	2.06	...
Slagged.	82.14	12.37	0.99	0.39	0.37	3.74	...
Siemens-Martin silica brick:							
Kernel..	94.67	1.84	1.00	1.67	0.46	0.36	...
Slagged.	90.66	2.94	2.50	2.87	0.60	0.43	...

An inspection of these typical analyses shows in almost every case an increase in the alkalies, coming mostly from the ash of the fuel used; in coke ovens, however, common salt in the wash water used for washing the coal may be an active agent in slagging the bricks, the chlorine going into the gases as ammonium chloride. Lime and magnesia increase somewhat in blast-furnace bricks, but more in open-hearth bricks, where they are subjected to the dust of the lime and burned dolomite shoveled in. The iron oxide increases somewhat in open-hearth furnace bricks, more in blast-furnace bricks, nearly as much in some bricks at the bottom of coke-oven retorts, but most in the cupola lining, where the oxidizing influences are stronger.

The silica decreases in every case, which phenomenon Dr. Blasberg surmises may be due to volatilization; the writer quite agrees with this explanation, and would add that the removal of silica by volatilization decreases the ratio of silica remaining to the other materials present, and, therefore, increases the fusibility. This change takes place even at moderate temperatures far below those prevailing in the Siemens furnace. Dinas bricks sometimes become dark gray; an analysis of such a slagged and discolored brick showed 0.65 per cent of carbon, probably deposited from hydrocarbon gases.

The Latest Development of Coke Ovens and Coking Practice was handled very extensively by Prof. FR. HERBST, of Aachen, in a communication covering 35 large quarto pages. In general we note that, in Germany, the coke-oven gas engine is winning its way against the steam turbine for power purposes; further that the producing of lighting and heating gases for cities is becoming more and more of a question of putting up coking plants which are adapted to giving the largest possible quantity of coking gas, thus giving great impetus to the regenerative oven and to the heating of the same by producer gas, usually from a bank of centrally placed producers.

The latest ovens are mostly of the type with vertically placed flues between the ovens, usually worked in four groups with central chimney or regenerator flue. The inclined ovens are also greatly in favor, with the retort at about 45°, since they allow of easy automatic charging and discharge themselves when the lower door is opened. Oven doors with gas-tight planed and water-cooled surfaces are being preferred to those luted with clay, being tighter, saving the clay and the labor of sealing up the retorts.

The newest Otto retort has gas supplied by 16 large bunsen burners, each supplying two vertical flues. The new Collin retort has a waste-gas flue between each pair of vertical heating flues. Brunck lays stress on distributing the gas to various places along the length of the retort, and has devised a new recuperator with shelf-formed heating surfaces and horizontal flues.

The oven of Kros, built by Salau & Birkholz, in Essen, has each vertical heating flue supplied with its own gas nozzle, with a horizontal flue along the whole length of the floor of the retort and thence communicating with the regenerators still further down beneath the retorts. The reversing of the draft is done entirely beneath the retorts. Each side flue is connected with two regenerative chambers, so that, as the retorts are charged alternately, the flues are always properly heated. Five of these ovens were run very successfully at Waltenscheid, reducing the coking period from 44 to 28 hours.

The Rheinischen Chamotte & Dinas-Werke, of Cologne, has likewise attempted the more uniform heating of the retorts by using burners of a particular construction entering both from beneath and from above, and arranged so as to give the most intimate possible mixture of gas and air. The variations in temperature on reversal are also reduced to a minimum. With these improvements, retorts up to 2.5 m (8 ft.) in height are in successful use.

The firm of E. Coppée, of Brussels, has also improved its regenerative oven, dividing the vertical heating flues into 10 groups of three flues each, and reversing the flow for regenerative purposes in each alternate group.

For city gas supply purposes the inclined retort with independent producer-gas supply for heating is making great advances. The gas supplied must be of at least 5000 calories per cubic meter, and such plants are, therefore, run so as to furnish the maximum of oven gas. If the coal gives gas averaging over 5000 (at Innsbruck 1 ton of coal gives 328 cu. m of gas of 5887 calories heating value), it is possible to increase the quantity of gas by sending in steam just before emptying the retort, in which manner as much as 400 cu. m of illuminating gas of over 5000 calories value has been obtained from 1 ton of coal. In such cases, with a production of 70 per cent of coke, about 25 per cent of the coke is used in gas producers to furnish the heating gas. Fig. 4 shows the newer ovens of this type as erected for the city of Munich by the Stettiner Chamottefabrik, on the Ries pattern. Each retort holds 6 tons of coal; the whole plant supplies 300,000 cu. m (10,000,000 cu. ft.) of gas daily. No trouble is met from retort carbon deposition in these furnaces. The lateral flues are also inclined.

The Vienna gas plant, now being built, is operated likewise by producer gas for heating the retorts. The producers use 13 per cent to 14 per cent of all the coke produced, and the evaporator under the producers, warmed by the hot gas, sends 0.8 kg of water into the producer for every kilogram of coke burned therein. The producer gas contains 21 per cent CO, 19.6 per cent H^2 and 10 per cent CO^2. The plant is partly horizontal ovens and partly inclined; its total capacity, in 1911, will be 1,000,000 cu. m (30,000,000 cu. ft.) daily.

In the new horizontal ovens, built after the Kopper's system, regenerators are provided for both heating gas and air, since these are of nearly equal volume. The arrangement of these beneath the retorts is very instructive (Figs. 5 and 6), two gas regenerators and two air regenerators being built together as shown, with considerable advantages.

The automatic machinery (Fig. 7) for bringing up the coal, stamping it into forms which have the shape of the oven, and charging it at once without manual labor have beeen greatly improved and are being rapidly adopted everywhere. These traveling loaders are highly developed, and described in detail in the article, as are also various minor details, such as the improved gas-tight doors, reversing valves, charge levelers, quenching devices, etc.

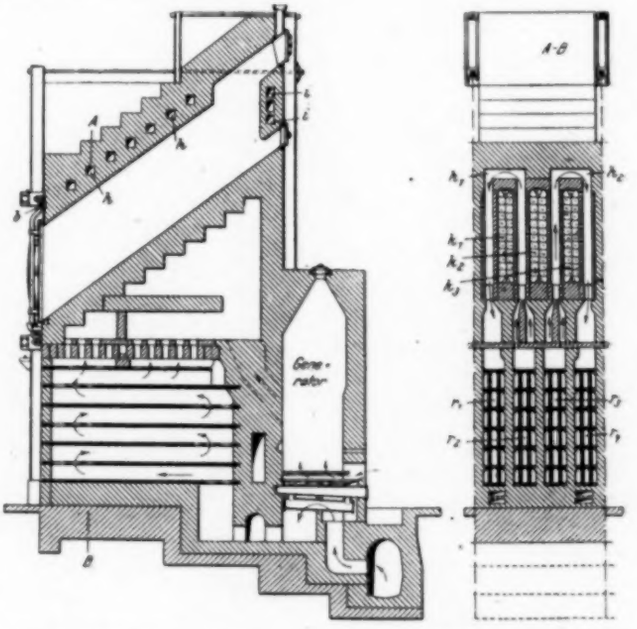


FIG. 4.—SECTIONS OF RIES COKE OVENS OF STETTINER CHAMOTTE FABRIK (*hi*, HEATING CHANNELS; k_1, k_2, k_3, RETORTS; r_1, r_2, r_3, r_4, REGENERATORS)

Progress in Recovery of Coke-Oven By-Products was treated by Prof. Dr. O. Rau, of Aachen, in a 28-page memoir containing a wealth of important information. Of the coal raised in the respective countries, the following percentages are coked:

	Germany.	Belgium.	England.	U. S. A.
In coking plants without saving of by-products	4	5	10	10
In coking plants with saving of by-products	16	11	2	2
In gas works with saving of by-products	4	4	6	1
Total percentage coked	24	20	18	13
Coked with saving of by-products	20	15	8	3
Percentage of coke made with saving of by-products	83	75	44	23

The world's production of by-product ammonium sulphate has risen to 500,000 tons per year, and of benzol to 90,000 tons. Investigations have shown that tar is formed between 200° C. and 500° C., and ammonia between 500° C. and 700° C.; by very slow distillation between the latter temperatures the recovery of ammonia may be increased by one-half. The tar is finding increased use in street paving and benzol in motor cars.

The waste gases are being used on a large scale for light, heat and power, particularly in large cities; coke-oven gas is piped 63 km (38 miles) to Barmen and sold there at 2.5 pfennig per cubic meter (18 cents per 1000 cu. ft.). The paper contains a lengthy discussion of the various reactions in the ovens at different temperatures, and closes with a complete bibliography of recent information on the subject.

Gas-Producers, by Ober-Ingenieur J. Hofmann, of Witkowitz, treated of the oldest and the latest forms of generators. In 1841 Ebelman proposed his shaft generator, with tuyeres, addition of iron slags as flux, and tapping out of the slag. Since 1879 there have been in constant use in Witkowitz producers without a grate, using lime as flux for the ash of the fuel, and tapping out liquid slag every one to two hours. Meanwhile, all the various types of grate and revolving-bottom

producers have been developed. Numerous analyses of coals and gas used and made in various producers in Europe are given.

Quite lately, however, a return has been made to the slagging shaft-furnace producer. (Fig. 8.) The firm of Sepulchre, Fichet & Heurty, of Paris, build and operate the S. F. H. generator, which is built like a small charcoal blast furnace, burns coke with addition of lime, sand and granulated blast-furnace slag for flux, and uses three to six water-cooled tuyeres, with blast at 0.5 m to 1.5 m water-column pressure. No steam is blown in.

When the coke ash contains iron or iron is in the fluxes, pig iron runs out with the slag tapped every one to two hours. Tests in Girancourt, near Nancy, with poor fuel, gave the following results:

Brown Coal.		Gas (Dry).		Slag.		Pig Iron.	
C	44.41	CO	28.3	SiO^2	42.17	C	2.30
N	0.45	H^2	7.7	Al^2O^3	18.83	Si	1.73
H	1.36	CH^4	7.5	CaO	20.57	P	0.23
S	6.50	CO^2	1.4	MgO	12.89	S	1.97
Ash	37.66	N^2	55.1	MnO	0.17		
H^2O	10.00			S	1.72		
				P^2O^5	0.04		
				FeO	1.26		

The brickwork of the crucible needs renewing every three months. The gas-producing power is high. A furnace only 1.2 m in diameter at the tuyeres and 1.13 sq. m area gasified 24.4 tons of fuel per 24 hours, or 900 kg per hour per square meter of section at the tuyeres (175 lb. per square foot). In actual amount gasified this is equal to the output of the ordinary 3-m (10-ft.) diameter producer.

Either in the slagging or the non-slagging producer it is recommended with caking coals to charge unburned limestone as a flux; this keeps the coal from caking too hard, the CO^2 driven off is reduced to CO and increases the value of the gas, while the lime forms a better slag or ash.

The author dwells on the injurious influence of water vapor in producer gas, saying it is impossible to produce best quality steel in the open-hearth furnace if the producer gas contains over 30 grams of moisture per cubic meter. The best way to achieve this is to superheat the steam used before sending it into the producer; this is *very strongly* recommended. It can be easily done by passing it through a coil heated by the producer gas itself. By doing this the producer gasifies more coal and gives better gas.

The making of producer gas has come to a turning point. It is now possible to utilize very poor fuel, such as is often thrown on the waste heap at the mines, to produce from it good gas and to completely burn it to ash or slag. In June, 1910, the Compagnie des Mines Blancy, at Montceau, will begin to gasify culm-heap residue containing only 20 per cent carbon, 15 per cent volatile matter and 65 per cent ash. Coke breeze is mixed with this material, and thus utilized. The complete utilization of our coal supplies is only possible by the manufacture and use of producer gas, which is, therefore, of highest economic importance.

The Production of Tar-Free Producer Gas was discussed by Dr. H. BRAUNE and EDV. HUBENDICK, of Stockholm. The authors start from the classification of hydrocarbons into aliphatic and aromatic compounds, the former further into paraffins (C_nH_{2n+2}), olefins (C_nH_{2n}) and acetylenes (C_nH_{2n-2}) all built upon the compound CH_4, and the latter built upon the closed benzol ring (C_6H_6). The paraffins are most resistant to heat, being able to withstand high temperature, such as 400° C. to 500° C. for hours.

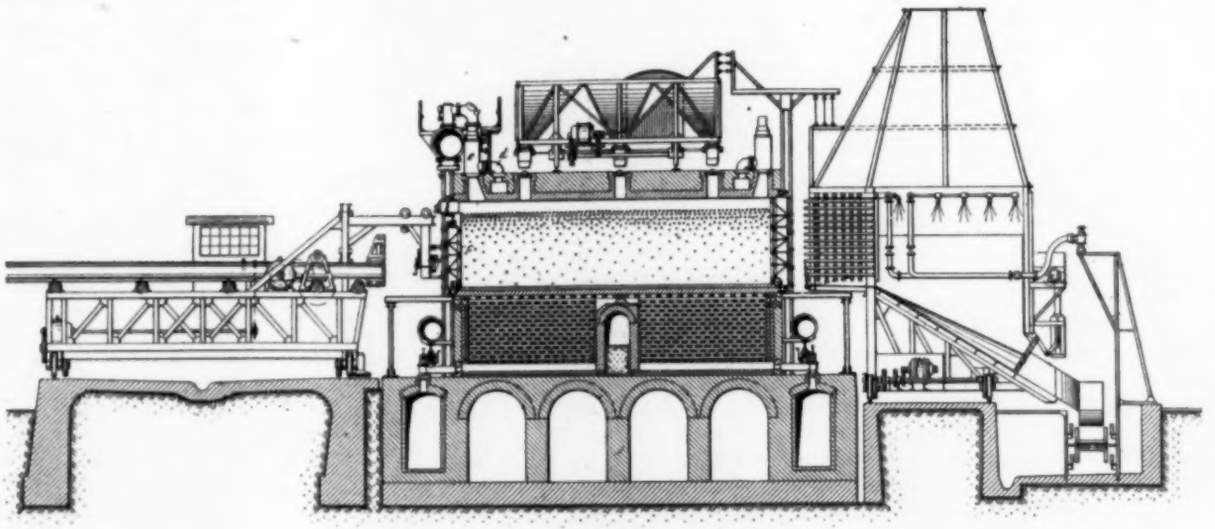


FIG. 5.—LONGITUDINAL SECTION OF KOPPER'S COKE OVENS.

Wood is a mixture of aliphatic and aromatic substances, the most important being the alipathic material cellulose. On dry distillation, on slow heating many condensable products result because of the oxidizing action of the water of constitution upon the hydrocarbons; on slower heating more gaseous products result.

Peat contains very little aromatic compounds, which is established by the nature of the dry distillation products.

Lignite, or *brown coal*, consists principally of alipathic compounds, yet there are found with the peat deposits aromatic compounds coming from the decomposition of resin. Such deposits are seldom found, but when found are of considerable value.

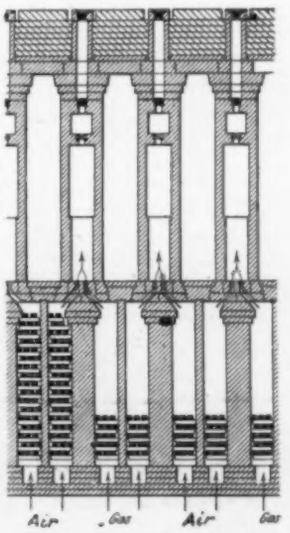


FIG. 6.—VERTICAL SECTION OF KOPPER'S COKE OVEN.

Bituminous coal consisted at one time principally of alipathic compounds, but by dry distillation gives the whole aromatic series; older coals contain more of the aromatic compounds.

In the gas producer aromatic compounds are weak and are easily decomposed at 750° C., the aliphatic compounds are not so easily broken up. It is, therefore, important, in making gas from coal which gives aromatic compounds, to keep the temperature of the issuing gases between 750° and 780°, and to

work with a large excess of hydrogen in the producer gas made by using steam. This prevents the decomposition into sooty carbon and hydrogen, and yet breaks up easily condensable aromatic compounds.

The aliphatic compounds from brown coal, wood, etc., are not to be thus decomposed, but require a temperature of over 1000° and oxidizing conditions. Aromatic distillation products should, therefore, pass through the fuel in the upper part of the producer, in presence of much hydrogen in the gas, at temperatures not over 750° to decompose them in a reducing atmosphere and so free the gas from tar; aliphatic distillation products are best drawn off and passed through the bottom of the producer with the air for combustion, so as to decompose them at a high temperature in an oxidizing atmosphere.

The Status of Iron-Ore Briquetting and Agglomeration in Germany was considered by Geheinier Bergrat G. Franke, of Berlin. Under iron ore were grouped also flue dust, pyrites, cinders, converter dust, roll scale and similar ferruginous wastes. There are at present nine briquetting plants, daily output 2200 tons and one agglomerating plant of 130 tons capacity. Together, they treat 500,000 tons of flue dust per year, averaging 35 per cent iron, which is about one-third of all which is produced by the German blast furnaces. Six new plants are under construction, and the time is not far off when all the flue dust will be briquetted.

The magnesium chloride process uses 1½ per cent of a waste 33 per cent $MgCl^2$ solution from the potassium chloride works, costing 26 pfennig (6½ cents) per ton of ore; the plant is cheap, 60,000 marks ($15,000) will build a plant treating 60 tons in 10 hours; the solution is mixed with the ore, pressed at 400 atmospheres, and let stand 24 hours. The total cost is 1.60 marks ($0.40) per ton of briquettes.

The lime process uses 6 per cent to 8 per cent of slaked lime, ground up with 1 per cent of blast-furnace slag; the moist mixture is pressed under 200 to 400 atmospheres, but the briquettes must stand several weeks in the air in order, by slow absorption of CO^2, to become sufficiently hard and strong, making the cost of this method high.

The Ronay process uses no binder, but presses the moist material at 1000 atmospheres and then bakes the briquettes at

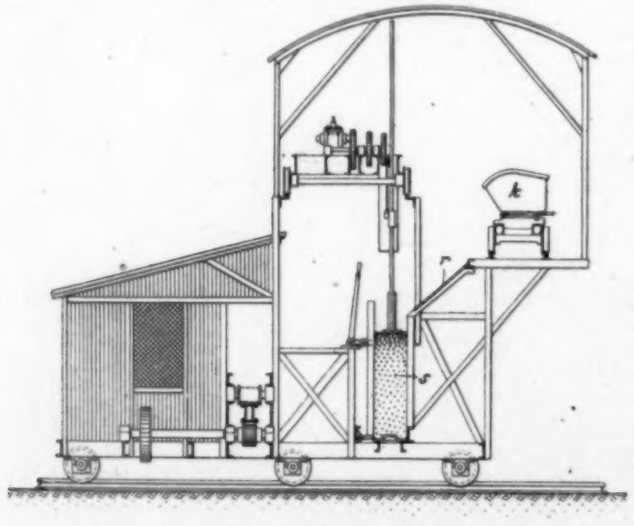


FIG. 7.—AUTOMATIC MACHINE (*k*, TILTING CAR; *r*, INCLINED PLATFORM; *s*, STAMPING FORMER).

low redness in a furnace heated by blast-furnace waste gas. It requires a comparatively expensive plant and costs much for power, still some new plants on this line are projected.

Only one plant in Germany sinters in revolving cylinders, firing with coal dust. The Giessener Braunsteinwerken vorm. Fernie, at Giessen, sinters Fernie manganese ore containing 20.5 per cent Mn and 22.5 per cent Fe, mixed with spathic ore containing some pyrites and zinc blende. The furnace plant is very dear, and the costs of treatment run up to 3 marks to 4 marks ($0.75 to $1) per ton of product.

The briquetting of turnings, filings and planings from iron and steel works by the Ronay method is very successful. Up to 2000 atmospheres pressure is gradually applied, and the briquettes thus made melt down with only 4 per cent to 10 per cent loss. They are particularly suitable for open-hearth and

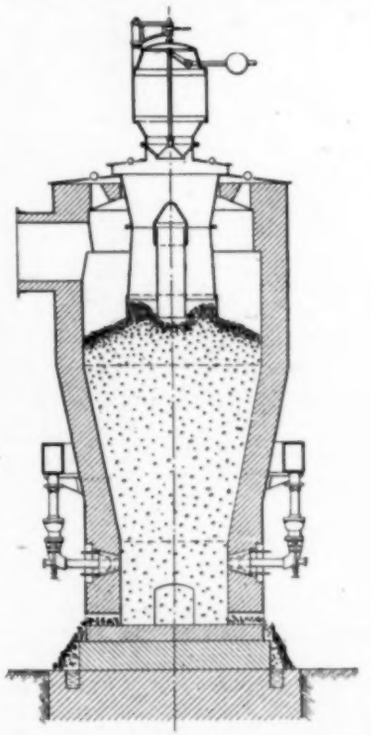


FIG. 8.—SLAGGING SHAFT-FURNACE GAS-PRODUCER.

electric steel manufacture, and are coming rapidly into general use.

Turbo-Compressors for Blast Furnaces were discussed by Prof. Dr. Ing. Rateau, of Paris. The first multiple-wheel, high-pressure turbo-compressor was started at Bethune in 1905. Turbo-blowers have been furnished to the blast furnaces of Vizcaya in Spain and at Chiers; very high-pressure blowers were constructed for the Rand Mines in the Transvaal, which have attained 65 per cent efficiency on isothermic compression. Two centrifugal compressors are being built for the Bessemer converters of Denain and Auzin. Over 80 of these high-pressure compressors are now in operation.

Hydraulic Air Compressors were described by Oberingenieur P. Bernstein, of Cologne. They produce a high degree of air compression with a small height of fall. Comparison with mechanical compressors shows much greater economy for the hydraulic. Besides using only water-power, the hydraulic compressor is very simple, reliable and always ready to work. The air supplied is free from dust and oil, and because of its coolness is low in moisture.

Reduction by Carbon and Carbon Monoxide was the subject of a short communication by G. Charpy, of Montluçon. During several hours heating at 1000°, in a vacuum, neither sugar carbon, graphite nor diamond carburized iron; the least trace of gas caused immediate cementation. CO gas carburizes iron from 600° up to 800° C.; it oxidizes chromium and manganese at 1000° and over, forming a deposit of carbon. Solid carbon does not reduce iron oxide, in a vacuum, until 950° is reached. CO gas commences to reduce Fe^3O^4 to metallic iron at 280°, but very slowly. NiO is similarly reduced.

Gruner's Ideal Working of the Blast Furnace, by Prof Dr. Jos. W. Richards, of Lehigh University. This paper has

been printed in full on page 403 of the July issue of this journal.

The Advantages of Hot Blast and Dried Blast in Blast Furnaces was discussed by Geh. Prof. Dr. F. WÜST, of Aachen. An interesting table is given concerning the changes produced by heating the blast, gathered from the literature of the blast furnace. The figures are quite curious, varying from 0 per cent fuel saving and 40 per cent faster running to 0 per cent faster running and 20 per cent fuel saving. If we arbitrarily add the two percentages together, there is obtained a minimum of 17 per cent advantage and a maximum of 98 per cent. Dr. Wüst concludes this part of his review by the statement that "although 80 years have gone by since hot blast was introduced, yet a satisfactory explanation of its action, free from all objections, has not yet been found."

The use of dried blast by Gayley led to the same extraordinary results; the saving was several times as great as could be calculated from the heat saved by removing the moisture. The results show from 11 per cent fuel saving and 21 per cent faster running, to 2 per cent faster running and 19 per cent fuel saving. Arbitrarily adding these percentages together, we have a minimum of 17 per cent advantage in September and November, and a maximum of 65 per cent in April. After reviewing all the proposed explanations which have been advanced (except that of J. W. Richards in *Trans.* Am. Inst. Mining Engineers, February, 1905, which was overlooked and not mentioned) Dr. Wüst again concludes that "all the explanations advanced have been insufficient and unconvincing."

He then offers his own explanation, as follows: When the blast meets carbon, there is first formed CO^2 and next CO. The velocity of these reactions depends on the temperature, the higher the temperature, caused by heating or drying the blast, the quicker these reactions occur, and the smaller the zone or area in which free oxygen or CO^2 exists at the tuyere level. As the reduced iron falls through this tuyere section, the part of it which drops through the space where CO^2 or O^2 exists is oxidized or refined, losing some of its silicon, manganese, fosforus and carbon, while some of the FeO dissolves in the slag.

These reactions require heat (according to Dr. Wüst's argument) and therefore the smaller this oxidizing zone the less heat is thus lost and the faster the furnace works, with less fuel.

The concluding part of Professor Wüst's explanation is neither very clear nor convincing. He concludes by predicting that blast enriched in oxygen (Lindeluft) would be another means for contracting the "oxidizing zone" and producing advantageous running in a way similar to hot blast and dried blast.

The Rationale of Dried Blast was further discussed by Prof. Dr. Jos. W. Richards, of Lehigh University. This paper was as follows:

The higher maximum temperature of the furnace gases in the smelting zone (region of the tuyeres) is the sole and the sufficient reason for the advantages produced by drying the blast.

Qualitative Considerations.—Removing the moisture of the blast results in saving the heat formerly required for its decomposition and enables the furnace gases to attain a higher maximum temperature of combustion. The higher maximum temperature of the furnace gases quickens the rate of transfer of the necessary smelting heat to the slag and pig iron, and thus increases the rate of smelting.

The smelting down of slag and pig iron to the liquid and superheated state requires to have added to those materials, at a temperature at and above their melting points, their latent heats of fusion and the sensible heat of superheated liquid material. This is the principal thermal work done in the smelting zone of the furnace. It can only be accomplished by the agency of furnace gases possessing maximum temperatures far above the melting points of these materials, and is accomplished only by the sensible heat which these gases are able to give out in cooling down from their maximum temperature to the melting points of the slag and iron.

The sensible heat which the furnace gases possess above the melting points of the slag and iron is therefore the only effective smelting heat which they possess, and is a direct measure of their smelting capacity. Any rise in the maximum temperature of the furnace gases increases in a large proportion the part of their sensible heat available above this necessary smelting temperature, and thus increases largely their smelting power.

When the smelting power of the carbon burned in the tuyere region is increased, more iron and slag can be smelted per unit of carbon there oxidized, and therefore the fuel consumption in the smelting zone is greatly decreased per unit of pig iron produced. This decrease of carbon consumption in the smelting zone does not necessitate the use of more carbon for reducing purposes in the upper part of the furnace, because there is ordinarily produced in the smelting region by the oxygen of the blast such a large excess of CO gas that only a small fraction of it can be possibly used up in reducing the ore. The poor ratio of CO^2 to CO in the gases of the ordinary furnace is not due to imperfect reduction of ore by CO, but to the fact that such a great excess of CO gas is formed in the tuyere region that the oxygen of the ore can only convert a small fraction of this CO into CO^2.

The smaller amount of carbon consumed in the tuyere zone when using dried blast is therefore more perfectly changed into CO^2 in the upper part of the furnace, during the reduction of the ore, and therefore gives greater thermal service in the furnace per unit of carbon consumed. This improvement in the ratio of CO^2 to CO in the throat gases is a direct result of the smaller carbon requirement in the smelting region.

The smaller amount of carbon consumed per unit of pig iron or of burden charged, results in a smaller volume of CO and N^2 gases, rising through the furnace per unit of burden. They therefore perform reduction more thoroughly, and become cooler before leaving the furnace. While, therefore, the smelting zone is working hotter, the upper part of the furnace works cooler.

Since the furnace works faster, the radiation and the conduction losses per day being no higher or even less than before, these losses expressed per unit of output become lessened.

Since less blast is required per unit of product, the stoves can heat it hotter and the blowing engines have less work to do, unless the blast is kept constant and the output increased to its maximum. In any case, the blowing engine power per unit of output is decreased.

Besides these direct results of the higher temperature in the tuyere region there are secondary advantages of removing or rendering uniformly low the moisture in the blast, which are of great value to the management of the furnace. These are the regularity of running, of output, of fuel consumption and of grade of pig iron produced. The manager does not have to fear sudden changes in the quality of iron produced, nor to carry an excess of fuel in the furnace charge in order to provide against such unforeseen changes. This regularity gives such great relief to the manager that the practical man may well count it as the chief advantage, as far as he is concerned, of using the dried blast.

But, uniformity and regularity are only one side of the advantages of dried blast; the other side is the economy resulting directly from the absence of high moisture and the consequent high temperature in the tuyere region. *Regular and uniform running at high efficiency* is the combination which in its entirety has made such a great advance in practical blast furnace running with dried blast.

Quantitative Considerations.—With dry air blown upon incandescent carbon, the maximum temperature of the gaseous products of combustion, CO and N^2, will be 1678° for cold

blast, and 84° hotter for every 100° that the blast is heated. With moisture present in the blast, these temperatures may be reduced as much as 235° when the air is warm and saturated with moisture, giving a theoretical maximum temperature of only 1443° with unheated blast. Details of these calculations, and tabulation of temperatures for various atmospheric conditions and temperatures of blast may be found in the writer's *Metallurgical Calculations*, Part II, pp. 282-286.

As an illustration, if the outside air carries 40 grams of moisture per cubic meter and the blast is heated to 500°, the maximum temperature of the gases of combustion in the tuyere region is 1848°; while if the moisture is reduced to 5 grams, and the blast temperature is unchanged, the maximum gas temperature rises to 2061°.

Assuming a smelting temperature of 1250° as required while melting is being performed, the first case gives an excess, or temperature head, of 598°, available to transfer heat to the charge as it is being smelted; the second case gives an excess of 811°, or an increase of temperature head of 213°, or 36 per cent. Under these conditions, the furnace would smelt down charge approximately 36 per cent faster with the dried blast than with the moist. Admittedly, these atmospheric conditions are extremes. The average increase on the Atlantic Coast of the United States has been 15 to 20 per cent for the year's running.

The three blast furnaces producing the largest number of tons of pig iron in 1909, in the United States, *were all three running on dried blast.*

The increased smelting power per unit of carbon burned in the tuyere region, when using dried blast, is also calculable. When a kilogram of carbon is burned to CO, it evolves 2430 calories; if the blast necessary is preheated, there is added to this the sensible heat in the blast. The total available smelting heat per kilogram of carbon, using dry-air blast, can be thus calculated, e. g.:

Temperature of Blast.	Available Heat.
0°	2430 Cal.
300°	2847 Cal.
600°	3286 Cal.
900°	3746 Cal.

If, however, moisture is present, it rapidly absorbs heat, and reduces the available heat in the smelting region. Its effect is therefore to leave less heat available at a lower initial maximum temperature. Taking again the example of blast at 500° and air with 40 grams of moisture to the cubic meter, the total heat available is 3137 calories from combustion and the blast, diminished by 574 required to decompose the water vapor, leaving only 2563 calories available. The calories available for smelting purposes in the tuyere region is thus decreased 18 per cent by the presence of the moisture. Remove this moisture, and the weight of carbon necessary to be burned in the tuyere region to do a given amount of smelting work will be reduced 18 per cent, i.e., the fuel requirement of the furnace will be reduced at least 18 per cent, calculating per unit of output. Other incidental advantages will increase the saving even above the calculated 18 per cent.

If less carbon is burned in the tuyere region per unit of output, the oxygen from the solid charges given off by reduction, and which is fixed in amount, will produce a relatively greater proportion of CO^2 in the issuing gases. This means that a greater proportion of the CO formed will be converted into CO^2, and therefore a greater utilization in the furnace of the calorific power of the carbon of the fuel. For example, if in a given furnace 75.3 kg of carbon is burned at the tuyeres to CO, producing there 175 kg of CO, and the solid charges give up in being reduced 41.8 kg of oxygen, this can convert 73.15 kg of CO into CO^2, and the total heat of oxidation of the carbon to this proportion of CO and CO^2 can be calculated as 6119 calories per kilogram of fixed carbon consumed. If by using dried blast, 18 per cent less carbon is thus consumed (giving, however, just as much available heat in the tuyere region) and 61.75 kg is burned to CO, producing 144.1 kg of CO, 73.15 kg of this will be likewise converted into CO^2 by the 41.8 kg of oxygen from reduction (producing just the same amount of heat in the upper part of the furnace) and the total heat of oxidation of the carbon to this proportion of CO and CO^2 can be calculated as 6928 calories. The carbon burned in the furnace is therefore utilized with some 13 per cent greater thermal efficiency when using the smaller quantity in the tuyere region required by dried blast.

Similar calculations could be made as to the smaller velocity with which the smaller amount of cooler gas passes through the upper part of the furnace, and the consequent better utilization of its reducing power and of its sensible heat in heating the descending charges. Analogous calculations could be made showing quantitatively the smaller radiation and conduction loss per unit of product, when using dried blast. The data for such calculations are, however, not as abundant or as trustworthy as those already used. A longer experience with dried blast in practice will enable these data to be also used quantitatively, with convincing results.

Resumé.—The drying of the blast increases the available heat, in calories, in the smelting region, per unit of carbon there consumed. This increase of available heat quantity increases correspondingly the maximum temperature of the gases in the smelting region. The increased quantity of heat, available at a higher initial temperature, increases in a multiplied proportion the smelting power of the furnace, i.e., increases the rate of smelting down in unit time and increases the quantity of smelting down done per unit of fuel. As secondary or incidental results of these favorable conditions, the furnace works cooler above, the proportion of CO^2 to CO in the gases is increased, the radiation and conduction losses are decreased per unit of output, and the furnace works with great regularity and uniformity. The given analysis explains satisfactorily the observed facts of practice, and calculations based on the above principles explain quantitatively, with satisfactory concordance, the observed economies and advantages.

Material and Heat Balance Sheet of the Blast Furnace, by W. G. Gillhausen, of Aachen. This was a very praiseworthy attempt to get accurate balance sheets of four furnaces, making respectively (I) 10 per cent spiegeleisen, (II) open-hearth iron, (III) Hematite iron, and (IV) basic Bessemer iron. The 40 large quarto pages give all the details, which may serve as the basis for a great many arguments and conclusions; they contain the results of an enormous amount of work, for instance, besides all the analyses of materials charged and pig iron produced, 40 to 60 gas analyses on each furnace, equally many determinations of heat in pig iron and in the slag. Test I lasted one month, test II three days, test III and IV each five days.

The best of the tests was undoubtedly the first, and we therefore reproduce the conclusions of this investigation.

Charged.		Flue Dust.	Slag.	Pig Iron.		Gases.	
Fe^2O^3	1113.99	5.25		Fe	843.23	O	363.28
FeO	137.78		2.64				
Mn^3O^4	184.77	6.00					
Mn^2O^3	0.32			Mn	105.70	O	50.60
MnO^2	24.65						
MnO	27.70		65.54				
CaO	328.25	14.84	284.25			O	8.60
MgO	81.55	2.83	78.72				
SiO^2	332.20	18.09	290.08	Si	24.03	O	12.75
Al^2O^3	71.04	5.59	65.45				
P^2O^5	1.67	0.20		P	0.85	O	0.06
SO^3	0.20	0.17				O	0.02
SO^2	9.74					O	4.86
S	1.10			Si	0.19		
CuO	1.75			Cu	2.00	O	0.35
CO^2	408.42					C	111.39
C	991.20	4.86		C	46.90	O	297.03
H	8.97					C	930.44
N	6.02					H	8.97
Ash	105.05	105.05				N	6.02
H^2O (chem.)	15.95					H^2O	234.66
H^2O (mech.)	218.71						
		59.23	891.73		1000.00		

The gases contained (dried) CO^2 11.40, CO 27.90, CH^4 2.59, H^2 2.30 and N^2 55.81 per cent. Temperature 350° C. The blast was at 740° mean barometer 748 mm, temperature of air 3.3°, moisture 9 grams per cubic meter. From these data, the blast per 1000 of pig iron was calculated as containing H^2O 30.66 kg, O^2 994 kg, N^2 3283 kg.

From these data, the following heat balance sheet was calculated:

RECEIVED (per 100 kg. of pig iron).

Combustion of carbon	328,676 Cal.	=	80.0%
Heat in blast	81,567 "	=	19.9%
Heat in charges	235 "	=	0.1%
	410,458 "	=	100.0%

DISTRIBUTION.

Reduction of iron oxides	148,551 Cal.	=	36.2%
" " silica	8,850 "	=	2.2%
" " manganese oxides	23,854 "	=	5.8%
" " phosphoric oxide	532 "	=	0.1%
" " copper oxide	45 "	=	0.0%
" " water (decomposed)	9,793 "	=	2.4%
(Total used in reductions)	(191,625) "	=	(46.7%)
Decomposition of hydrates	121 "	=	0.0%
" " carbonates	33,376 "	=	8.1%
Evaporation of water	16,638 "	=	4.1%
Sensible heat in dust	352 "	=	0.1%
" " " gases	53,718 "	=	13.1%
" " " slag	37,146 "	=	9.1%
" " " pig iron	26,500 "	=	6.5%
Radiation and conduction (difference)	50,982 "	=	12.3%

In connection with this balance sheet, which is the best of the four, the following criticisms are almost self-evident, and should be considered by anyone making use of the figures:

1. The latent heat of vaporization of the moisture in the blast should *not* be reckoned in, because it comes to the furnace already as vapor. If it is counted as available heat, then the heat necessary to decompose the moisture of the blast considered as *liquid* water should be used. It is a mistake to reckon it on one side and not on the other. This makes a difference of 1858 calories.

2. The volume of gases was incorrectly calculated, since the carbon in the flue dust was not subtracted. This made too much carbon calculated as consumed. This mistake makes a difference of 1643 calories.

3. The heat of formation of the slag from its constituent oxides is omitted altogether. This can be approximately calculated, but not accurately. It is better to calculate it than to leave it out altogether. It amounts approximately to 5300 calories.

4. The heat of formation of the pig iron from its constituents is omitted. This is not accurately known, but can be estimated in this case at 3306 calories.

5. The heat evolution is therefore probably about 5000 calories too low.

6. The moisture in the blast, outside air, at 3.3° C. and 748 mm. barometer, is given as 9 grams per cubic meter. This is at least double what it could possibly have been, and the quantity of moisture in the blast and heat absorbed in decomposing it is probably at least double what it should be given.

For various reasons, not easy to discern, tests II, III and IV were unreliable in that the heat balance sheets did not check. They left over for radiation and conduction only 5.08 per cent, 0.58 per cent and even —6.20 per cent, and it is very difficult to fix the unreliable factors. Test I has been given in detail because it seemed the most reliable of the four. We congratulate the author, Herr Gillhausen, on having done a splendid piece of work, entailing an enormous amount of time and effort, such as has probably not been accomplished since the classic researches of Sir Lowthian Bell, and we hope he will accept our comments on their small imperfections in the same spirit as they are made.

The Theoretical and Practical Importance of the Electric Pig Iron Furnace was discussed by Privatdozent Ingenieur CARL BRISKER, of Leoben. He considers the electric furnace as even now able to hold its own in places where power is cheap, and that works manufacturing special steels will soon be able to utilize them to great advantage for producing the specially pure pig iron which they need.

He studies in detail the reactions in the Domnarfvet furnace in Sweden, making the assumption that the iron oxides are reduced according to the following reactions:

$$Fe^2O^3 + 2C = 2Fe + CO + CO^2$$
$$Fe^3O^4 + 3C = 3Fe + 2CO + CO^2$$

This assumption is supported by analyses of the gases just above the smelting zone, where there was found 33.6 per cent CO^2 and 56.3 per cent CO in the gases, and there was calculated from the above reactions 37.4 per cent CO^2 and 54.5 per cent CO.

Further up, at the throat, the gases were found to be 41 per cent CO^2 and 45.6 per cent CO, which change the author ascribes to the splitting up of CO into C and CO^2 in the upper part of the furnace shaft. This would require the deposition of 1.73 kg C from the CO gas, and set free an amount of heat sufficient to warm the gases 337°, at that part of the furnace shaft where the reaction takes place. The author does not prove this by temperature measurements taken in the furnace.

He regards this carbon deposition as of great importance to the reduction in the electric shaft furnace, and as resulting in a much better ratio of CO^2 to CO in the issuing gases. The gases, to take advantage of this reaction, must be cooled to the necessary degree by passing through a tall shaft, else the reaction cannot take place. An interesting and instructive comparison of the heat balance sheet is made with an average coke blast furnace, as follows:

HEAT RECEIVED (per 100 kg. of pig iron).

	Electric Furnace.			Blast Furnace.		
Electric current	254,900 Cal.	=	70.8%			
Combustion of carbon	105,100 "	=	29.2%	237,500 Cal.	=	84.7%
Hot blast				43,200 "	=	15.3%
Totals	360,000 "	=	100.0%	280,700 "	=	100.0%

HEAT DISTRIBUTION.

For reductions	170,147 Cal.	=	47.2%	170,147 Cal.	=	60.6%
Heat in iron	26,500 "	=	7.4%	26,500 "	=	9.4%
Heat in slag	12,628 "	=	3.5%	16,100 "	=	5.7%
Decomps'n of moisture	6,496 "	=	1.8%	8,303 "	=	2.9%
Heat in gases	3,080 "	=	0.8%	20,000 "	=	7.1%
Cooling water	77,500 "	=	21.5%	19,000 "	=	6.8%
Radiation and conduction	63,649 "	=	17.8%	20,650 "	=	7.5%
Totals	360,000 "	=	100.0%	280,700 "	=	100.0%

The extraordinary outlay of nearly 40 per cent in the last two items in the electric furnace, as against 14.3 per cent for the same item in the larger coke furnaces leads to the conclusion that as soon as the electric furnace is run on anything like the scale of the blast furnace in question (200 tons per day), a saving of at least one-half of this loss will be made, amounting to a saving of 70,000 calories per 100 kg of pig iron, and since this will all count on the electric current used, it will mean a reduction from 254,900 to 184,900 calories required for the current, or a saving of 28 per cent in power; or, putting it in another way, a reduction from 0.46 hp-year per ton of pig iron to 0.33 hp-year; the reviewer believes it will even be reduced to 0.28 hp-year or less.

Microscopic Examination of Blast Furnace Slag for Cement Making is recommended by Dr. H. PASSOW, of Blankensee, as especially valuable in determining the value of the slag for the manufacture of Portland cement. Only the vitreous slags have a market value for cement making, and while chemical analysis shows no distinction whatever between vitreous and non-vitreous, the microscope must be constantly resorted to, since it shows at once whether granulation or crushing has yielded vitreous or non-vitreous slag particles.

When the microscope shows that granulation or crushing or the subsequent drying has yielded a too large proportion of non-vitreous particles, then suitable remedies must be applied at once to stop this action. Numerous data and micrographs were shown to substantiate this. (Figs. 9 and 10.)

The microscope is also very useful in controlling the qualities of new blast-furnace slags, the granulated or atomized slag can be examined at once, and the strength of air blast or water current varied to give the best vitreous product. Slags with SiO^2, 27 to 31 per cent; Al^2O^3, 13 per cent to 20 per cent, and CaS, 4 per cent to 8 per cent, are the best reacting slags

for cement manufacture, not needing nearly so fine grinding as those higher in SiO^2 and lower in Al^2O^3 and CaS. Such slags are somewhat more difficult to granulate, however; they must be quickly but uniformly cooled; the Buderuss Iron Works have arranged a trommel which does this very well. The slower reacting slags are easily vitrified, but require larger additions and more fine grinding to make good cement.

The author concludes that the slag-cement business can only

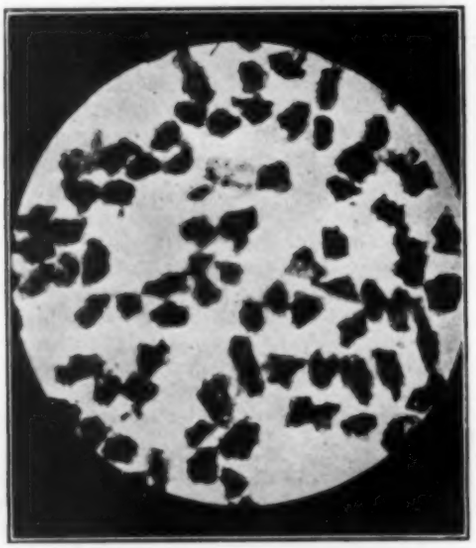


FIG. 9.—NON-VITRIFIED SLAG.

flourish by using the microscope freely as a constant guide in controlling the quality and granulation of the slag.

The Status of Cleaning Blast Furnace Gases in Germany was handled by Ober-Ingenieur C. Grosse, of Metz. The German blast furnaces produce 13,600,000 tons of pig iron yearly, corresponding to 7,650,000 cu. m of waste gas hourly, of which 4,840,000 cu. m, or 63 per cent, is being cleaned. The modern requirements are: (1) the gases must be cooled as

FIG. 10.—VITRIFIED SLAG.

much as possible to free them from water vapor; (2) the gas for heating purposes should be cleaned to 0.1 to 0.5 gram of dust per cubic meter; (3) the gas for gas-engine service should be cleaned to 0.01 to 0.03 grams of dust per cubic meter at 25° C.; (4) all apparatus should be simple and reliable. The cooler the gas is made the more moisture is removed and the easier it is to remove dust because of its smaller volume.

The four principal mechanical purifiers are the Zschocke, Theisen, Bean and Schwarz, which purify, respectively, 44 per cent, 33 per cent, 14 per cent and 5 per cent of all the gas cleaned. The Flössel (Fig. 11) has experimentally given better results than the Theisen. The proper cleaning of the gases causes a saving of 20 kg to 25 kg of fuel per ton of iron produced, the blast is usually heated 60° to 100° hotter, the heating-up periods of the stoves are reduced almost one-half, and the "on-blast" periods are lengthened half an hour. The stoves require on an average 42 per cent to 44 per cent of the whole gas cleaned, the blowing engine 16 per cent to 18 per cent, leaving 37 per cent to 42 per cent for outside use. Used under boilers, the cleaned gas gives 10 per cent more steam.

The average boiler campaign between cleanings is raised from one month to three or four months, the hot-blast stove campaign from two or three months to 12 to 14 months. The chimney gases from the Cowpers should not run over 300° C., but higher Cowpers are necessary with the cleaned gas because of its higher heating power; stoves up to 40 m (130 ft.) in height have been constructed with 6 m to 7 m (19 ft. to 23 ft.) diameter. The author thinks that in a short time all the blast-furnace gas produced in Germany will be properly cleaned and utilized.

Large Gas Engine Installations was treated by Prof. Langer, of Aachen. Improvements have been made in reducing the mean effective piston pressure from 5.5 kg to 4.5 kg per cubic centimeter (82 lb. to 67 lb. per square inch), with a regular total load factor of 70 per cent. The fundamental condition of satisfactory working was said to be "a high excess of

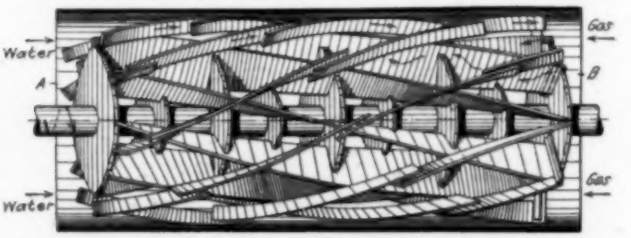


FIG. 11.—BLAST-FURNACE GAS PURIFIER OF FLOESSEL.

air in the mixture." This conclusion was not agreed to by everybody.

The Status of Foundry Casting in Germany was described by Ingenieur Irresberger, of Mülheim (Ruhr). The foundry is no longer neglected in Germany, but now employs technical graduates, special engineers, chemists, mechanical testing laboratories, costly mechanical outfit for molding, transporting, charging cupolas, etc. Great attention has been given lately to sand-blast cleaning of castings, the workman being protected by an air-shield. Many foundries run only on specialties. In Germany 120,000 hands are employed in 1560 foundries producing annually 2,000,000 tons of castings. The traveling cupola charging machines (Fig. 12), sand-blast machines, and other cleaning devices were described at length. Many foundries have a small laboratory simply to determine silicon in the iron. Low-carbon iron is increasingly used for strong castings; many foundries produce it by charging into the cupola Ronay briquettes of soft-steel and wrought-iron turnings.

Cores are often made of sand, meal and glutrose and baked in ovens heated by hot water to 250° C. to 300° C. Such cores are exceptionally resistant during casting, and later fall to powder. Molds are usually dried by burning gas; at some foundries cleaned blast-furnace gas has been used with best results. Blast-furnace foundries, which take their melted iron directly from the mixer, are increasing rapidly in numbers.

The Status of the Basic Bessemer Process in Germany was the subject of a splendid report by Superintendent Esser, of Differdingen. The production has risen from 329,000 tons in 1883 and 3,004,000 tons in 1896 to 7,668,600 tons in 1909. The number of converters running in the years named was 47,

70 and 108, and the converter charge averaged 9 tons, 12 tons and 17 tons, respectively. In general the original form of converter has been preserved, but the large new ones have a shallower bath; in the 15-ton converters the bath was 0.7 m to 0.75 m (28 in. to 30 in.) deep, but in the new 25-ton converters now in use it is only 0.45 m to 0.5 m (18 in. to 20 in.). This gives much less resistance for the blast to overcome, requires less blowing-machine power, and saves 1 per cent to 1½ per cent loss of iron. The Jünkerather Company are building a 30-ton converter which will be 6.8 m (22 ft.) high and weigh 70 tons without the charge and 100 tons when charged.

Pig-iron mixers of the rolling type are now in use up to 1000 tons capacity; hydraulic power is used on the converters, but electric motors on the mixers. Electric locomotives are in general use to transport the pig iron, provided with controllers for tilting the ladles, all under the hand of one man. For handling molten steel the author prefers electrically operated casting ladles, but electrically operated high-pressure pumps for raising and lowering the heavy ladles.

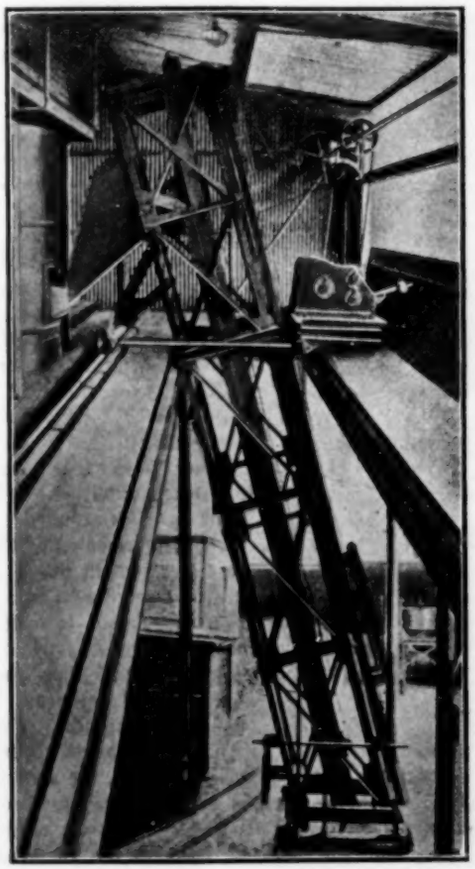


FIG. 12.—TRAVELING CUPOLA. CHARGING MACHINE.

For filling the converters and casting into molds overhead cranes are probably better than any other arrangement. Casting into moulds on trucks, from a ladle held by a traveling crane, has great advantages; the ingots thus cast are better than stationary ingots, because the bumping which they get while being moved brings to the surface gases, oxides and other impurities, and improves their quality. Lime is brought directly from the lime kiln to the converter mouth by a rope railway.

Fig. 13 shows the arrangement of the converter building in Rote Erde.

The converter bodies now stand about 280 charges, the bottoms when furnished with tuyere bricks 95 charges, if made with sample perforations 60-65 charges. This result has been attained mostly by the use of carefully chosen lining material, and above all by using compressed air stamps for ramming in the lining. The same are used for lining the steel ladles, and their service has been raised from 25 up to 50 or even 60 charges.

Where the bessemer works is far from the blast furnaces, it is advisable to run the pig iron into a mixer, and to cast only from the mixer. In this way a uniform quality of pig iron is furnished to the steel works. The great use of blast-furnace gases for power is driving the steel works to the ore regions, so that the iron industry is at present moving away from the coal to the ore.

The "Minette" deposits of Lothringen, Lorraine and Luxemburg are the chief source of the basic pig iron. Dr. Kohlman estimates that there is 5,000,000,000 tons of this ore. There are two principal varieties, the lime gangue variety and the silica gangue, which average about

	Lime Gangue.	Silica Gangue.
Iron	28-37%	32-40%
Lime	12-20%	7-9 %
Silica	6-9 %	12-22%
Phosphorous	0.6-0.7%	0.6-0.7%

The usual ore mixture is 2/3 calciferous to 1/3 siliceous ore. It appears as if there is Minette ore to last a century.

It has been said against the basic bessemer process that it is not possible to make by it steel low enough in phosphorus. The reason why phosphorus is not usually blown below 0.06 per cent is not that the waste of iron is so much greater, but that by this further oxidation the steel is made useless by taking up an unbearable amount of oxygen.

The most important consideration at this point of the process is not whether the steel contains 0.06 or 0.04 or even less phosphorus, but that it should be made with as little oxygen in it as possible; the question to be decided is to keep the absorption of oxygen down and to take out the oxygen which has been absorbed.

If rails are called for with less than 0.04 per cent phosphorus, one would not willingly try to make them in the basic bessemer; it cannot be denied that it might be possible to bring the phosphorous that low, but the danger of "over-oxidation" stands imperatively in the way. The author does not think that a 0.04 phosphorus rail made with this danger of over-oxidation would average any better quality than the ordinary 0.06 to 0.07 phosphorus rail made with the ordinary degree of oxidation; he does not believe that foreigners will continue long to pay 20 marks ($5) per ton extra for such low phosphorus open-hearth rails.

Basic bessemer metal is hottest at the end of the blow, and is always let stand several minutes before casting, in order to give time for the final additions to become well alloyed. As the last of the slag is poured off, the ferro-manganese is added; the charge is then allowed to rest in the converter 7 to 8 minutes, to give a good alloy, allow gases to rise, oxygen compounds to float up, and any other impurities. The thin cover of slag on the bath at this time keeps in the heat and keeps out the air from the bath. Working in this way, the steel is much improved and casts well.

Comparing the basic bessemer with the basic open-hearth process, the comparison is not as bad for the bessemer process as Dichman and some others have tried to make it. From 1000 tons of pig-iron the bessemer makes 890 tons of steel, at a cost of 49.76 marks ($12.44) per ton, while the open hearth makes at the most 1100 tons, at a cost of 51.80 marks ($12.95) per ton. The basic bessemer process has made extraordinary progress in economy and in quality of product, so that it can say, in the immortal words of Mark Twain: "The news of my death is highly exaggerated."

The Pig and Ore Process in Germany, by Direktor R. GENZMER, of the Julienhütte in Upper Silesia, may be considered as a counter-stroke to Superintendent Esser's paper on the basic bessemer process. It is a fine review of the status of this process at the present moment, in Germany.

It would be impossible to make steel with economy in the

open-hearth furnace by oxidizing pig iron by the furnace gases, but it is done economically and quickly by using iron ore as the oxidizing agent. When this is done in furnaces as large as 50 tons capacity, making 200 tons per day, it becomes an active competitor to the basic bessemer process. The Talbot process is not used in Germany, the Duplex process is scarcely known there, and the Monell process seems (under this name) not to be known to Herr Genzmer, so he divides the pig and ore processes into the following:

I. Pig and ore processes, using liquid pig iron below 1 per cent phosphorus:

(a) Taken direct from the blast furnace.

(b) Taken from an ordinary gas-heated mixer.

(c) Taken from a mixer used as a pre-oxidizer.

II. The Bertrand-Thiel or Hoesch process using liquid pig-iron with over 1 per cent phosphorus (basic bessemer pig).

The question of the "mixer" is a highly important one for the steel maker, particularly that of the gas-heated mixer used as a preliminary oxidizer, but many of the German works which have made great improvements in this line refuse to let their improvements be published.

The use of low phosphorus pig iron direct from the blast-furnace is practised at the Hubertushütte in Upper Silesia, the open-hearth furnaces being fired by coke-oven gas. Fifteen

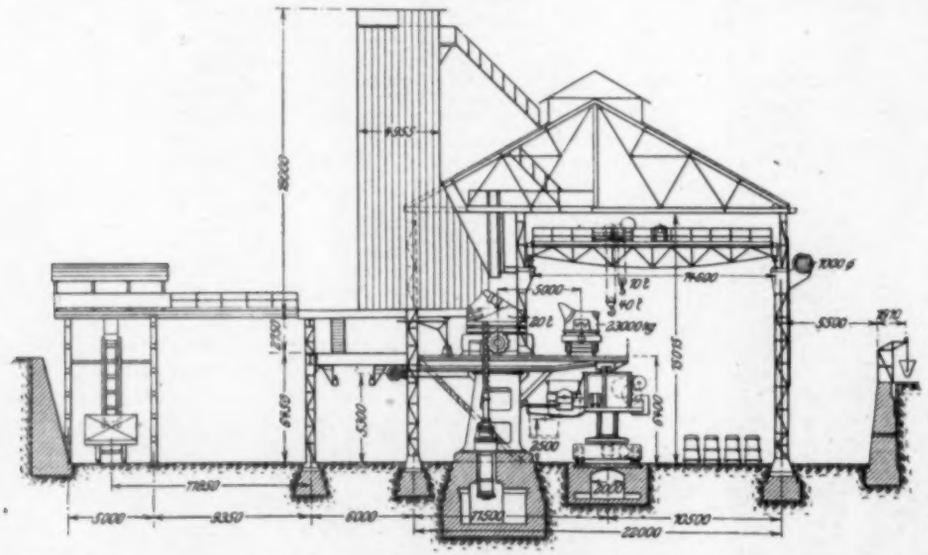


FIG. 13.—CONVERTER BUILDING IN ROTE ERDE.

tons of liquid iron is brought directly from the blast furnace to the 25-ton open hearth, and run into it from the bottom of the ladle.

This is not so easy a way as pouring from the ladle, but it has been found of advantage because the shaking during transport from the blast furnace seems to bring the sulphur (as manganese sulphide) to the surface of the metal, and by tapping from beneath this is left in the ladle. As much as 30 per cent of the sulphur is thus taken out of the pig-iron.

Ore and lime are charged into the furnace before the liquid pig is poured in; if the ore is somewhat preheated the reaction may be quite violent,* a foamy slag forms and the ore disappears inside of an hour. Analyses are given of the metal during a heat, in which 22.4 tons of pig iron was poured upon 4.7 tons of Swedish magnetic ore and 0.9 ton south Russian ore.

	C	Si	Mn	P	S
Pig iron used.....	3.61	1.89	2.320	0.670	0.044
After reaction (1 hour)	2.72	0.50	0.038	0.005	0.030
At tapping (6¼ hours)	0.087	0.004	0.499	0.067	0.042

The failure to remove the slag at the end of the reaction is the cause of the strong re-phosphorization, which at one time (4½ hours) amounted to 0.140 per cent. More recently, it is said, that this work is removing half the slag at the end of the reaction period, and since it contains 15 to 18 per cent of iron, the output is reduced about 1 per cent by so doing. The end slag carries 9 to 11 per cent FeO and a similar amount of MnO, with 5/6 per cent P^2O^5 and 20/25 per cent SiO^2.

The use of a gas-heated mixer is a great advantage. In it sulphur is always reduced, usually 20 to 30 per cent. Average analyses of pig iron and mixer metal, at the Julienhütte showed:

	Si.	Mn.	P.	S.
Pig Iron.....	1.93	2.03	..	0.092
Mixer Metal.....	1.43	1.72	0.43	0.063

The 150-ton mixer is electrically tilted, heated by hot producer gas, with regeneration for air only. Its temperature is kept between 1300° and 1400°C. The coal used is only 8 tons for 400 to 500 tons of pig-iron passed through per day. Only a small amount of slag is made, 1½ to 2½ tons per day, consisting of 27 to 35 per cent FeO, 29 to 37 per cent SiO^2, 13 to 18 per cent MnO, and 0.35 to 0.40 per cent P^2O^5. The hearth is magnesite, roof silica brick; it has run 1½ years without any stop, and repairs are exceedingly small.

In the furnace building, the furnaces of 40-ton capacity are served by overhead traveling cranes. Each furnace has its own producers, the total coal consumed is 20 per cent of the steel output, including mixer, Sunday heating, etc. Four charges are run per day. The charge is 70 to 80 per cent pig iron. At the end of the reaction period, part of the slag is run off, particularly when better quality steel is being made; the first slag carries 20 to 30 per cent Fe.

Mixers of 125 to 250 tons capacity used as oxidizers are in use at a number of German works. At the Georgmarienhütte (Fig. 14) the mixer is built like a Wellman furnace, but the practice differs from the Talbot furnace in that the oxidation is only carried a little way, and then the metal, which is still pig-metal, removed to the steel furuaces. The hearth is magnesite brick covered with burnt dolomite, the roof is Dinas brick. It is heated by purified blast-furnace gas, brought by a 0.8 meter diameter conduit. The same gas is used for the soaking pits, and will later be used in the Siemens-Martin furnaces. After the pig iron is run into the mixer, 5 to 6 per cent of its weight of iron ore and 3 to 4 per cent of lime is added. Per day about 4165 tons is put through this mixer: 354 tons of Swedish magnetite is added, producing 358 tons of slag, containing 6.86 per cent Fe. (24 tons altogether); 90 per cent of the iron of the ore added is reduced into the bath.

	C.	Si.	Mn.	P.	S.
Pig Iron.....	3.80	0.6-1.0	2.75	0.40	0.05
Mixer Metal.....	3.50	0.08	1.20	0.15	0.02

In the open-hearth furnaces, per day, 715 tons of ore was used, producing 2.24 per cent of reaction slag with 25.5 per cent Fe. and 7.18 per cent of end slag with 9.4 per cent Fe. The total slag production, including the mixer slag, was therefore 14.8 per cent. In the open-hearth reactions 83 per cent of the iron in the ore was reduced; of the total ore used, 85 per cent of its iron was reduced into the metal. One 50-ton furnace made 1464 tons of high-grade steel in 6 days.

The use of the mixer as a pre-oxidizer has these advantages: (1) Very small repair costs; (2) only two men to handle it; (3) only 2.65 per cent dolomite needed; (4) 103 to 104 per

*This is the pure and simple Monell process, if the slag formed is removed at the end of this reaction. [Translator.]

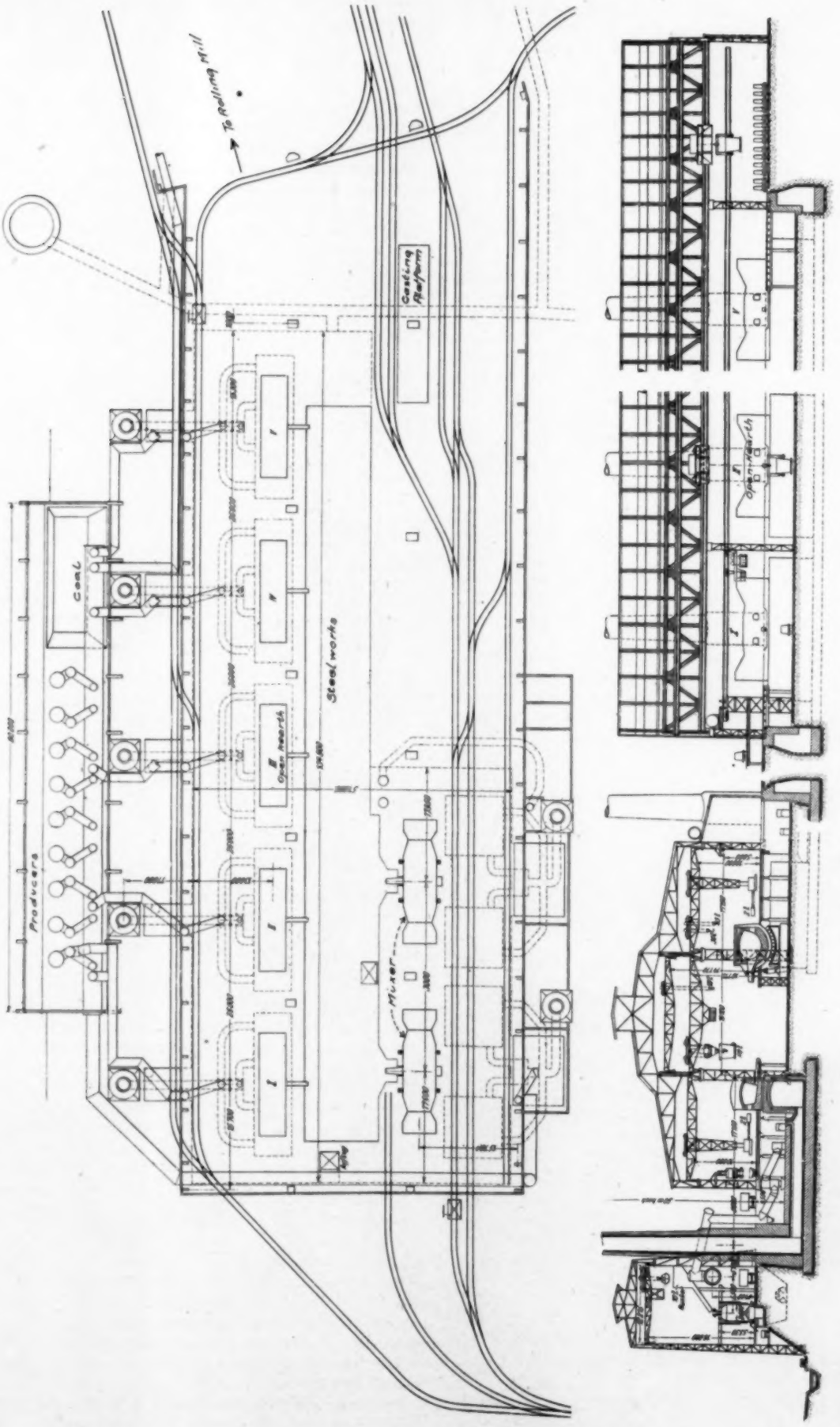


FIG. 14.—OPEN-HEARTH STEEL PLANT OF GEORGSMARIENHÜTTE.

cent output, reckoned on the pig iron; (5) only 16.3 per cent coal consumption in the open-hearth furnaces. It really serves as a preliminary open-hearth furnace; kept always at a moderate temperature, and doing part of the work of the open-hearth furnace while the metal is being kept melted as a reservoir.

The Hoesch process is being worked in 30-ton furnaces in Germany, and some 50-ton furnaces are being built. The pig iron is passed through a mixer, gas-fired, where it is not oxidized. So far, only basic bessemer pig iron, with over 1 per cent phosphorus, is thus worked. The furnace is charged with high phosphorus Swedish magnetite, roll cinder and burnt lime, and the molten pig run in upon it. A charge was, for instance, 3.44 tons Swedish ore, 0.77 tons roll scale, 1.88 tons lime, and on this 23.28 tons of pig iron. After the reaction was complete and the charge well heated, in 2 to 3 hours, the whole was tapped into a ladle and the slag run away. The furnace then received immediately a fresh charge of 2.18 tons spathic iron ore, 1.29 tons of lime and 5.412 tons of scrap, and the metal poured back into the furnace. After 1¼ hours heating the charge was well melted, all the ore dissolved and a good slag made; a further treatment of 1½ hours, with further additions of roll scale, lime and finally ferro-maganese, brought the metal to the tapping point.

	C.	Si.	Mn.	P.	S.
Pig Iron	3.28	0.32	0.96	1.86	0.132
After 1st Reaction	1.46	trace	0.34	0.26	0.082
After 2d Reaction	0.38	trace	0.29	0.09	0.100
Tapping	0.08	trace	0.47	0.04	0.067

Slag.

	SiO^2	CaO	P^2O^5	FeO	MnO
After 1st Reaction	11.40	48.86	22.13	4.67	3.93
After 2d Reaction	14.20	33.64	6.25	19.64	13.49
At Tapping	12.40	46.28	5.00	17.03	10.25

The dephosphorization is only certain, according to Springorum, when the slag has 40 to 45 per cent CaO and the SiO^2 is below 159.* The large amount of MnO in the slag tends to keep down oxidation of the bath, since the Mn reduced from it by the carbon of the metal tends to keep the bath deoxidized.

The original pig iron is low in manganese, but the spathic ore used in the second oxidation reaction is high in manganese, and thus provides the necessary MnO for the slag. The output is 104 per cent on the pig iron used; the first slag is high in soluble P^2O^5, and is kept separate and sold.

By thus working, the open-hearth process is able in Germany to compete with the basic bessemer (Thomas) process, using the same pig iron. With larger furnaces of 50 to 80 ton capacity the output is increased so greatly (an 85-ton furnace at Bochum produces 225 tons per day) that the open-hearth furnace is continually cheapening its product, and perhaps will eventually displace the basic bessemer altogether.

Dephosphorization of Iron in Presence of Carbon was discussed by E. Richarme, of Toaritsinsky-Tavod, Russia. His thesis is that 3 (Fe, Mn, Ca, Mg) $O.SiO^2$ has the same dephosphorizing power as 3 (Fe, Mn, Ca, Mg) $O.Al^2O^3$, or 4 (Fe, Mn, Ca, Mg) $O.P^2O^5$; or 2 (Fe, Mn, Ca, Mg) $O.Fe^2O^3$; or putting it another way, that 4 SiO^2, 4 Al^2O^3, 3 P^2O^5 and 2 Fe^2O^3 all have equal powers of keeping phosphorus *out* of the slag, and FeO, MnO, CaO and MgO have equal molecular powers of keeping phosphorus *in* the slag.

Starting with the calcium silicate as a type, 3 $CaO.SiO^2$, containing 26.3 per cent of SiO^2, the following are equivalent slags as far as dephosphorizing power is concerned: 3 $CaO.Al^2O^3$, 4 $CaO.P^2O^5$, 2 $CaO.Fe^2O^3$. The author then makes use of the molecular weights to get the equivalence of Al^2O^3, P^2O^5 and Fe^2O^3 in terms of SiO^2 and of FeO, MnO and MgO in terms of CaO. He then expresses the index number or "acidity" of the slag as the ratio of the summated acids to the summated acids plus summated bases. The finding of the index number is then simple: e.g.:

$$\%\ SiO^2 + \frac{\%\ Al^2O^3}{1.69} + \frac{\%\ P^2O^5}{1.77} + \frac{\%\ Fe^2O^3}{4.00} = \text{Summated acids}$$

$$\%\ CaO + \frac{\%\ MgO}{0.71} + \frac{\%\ FeO}{1.28} + \frac{\%\ MnO}{1.27} = \text{Summated bases}$$

Applying this formula to the actual slags obtained in the basic steel processes, he comes to the following conclusions, which if reliable are very important:

Slags with acidity ratio 0.32 to 0.26 will remove and retain phosphorus in an oxidizing atmosphere.

Slags with acidity ratio 0.26 to 0.22 will remove and retain phosphorus in an oxidizing or a neutral atmosphere.

Slags with acidity ratio below 0.22 will remove and retain phosphorus in an oxidizing, a neutral or a reducing atmosphere.

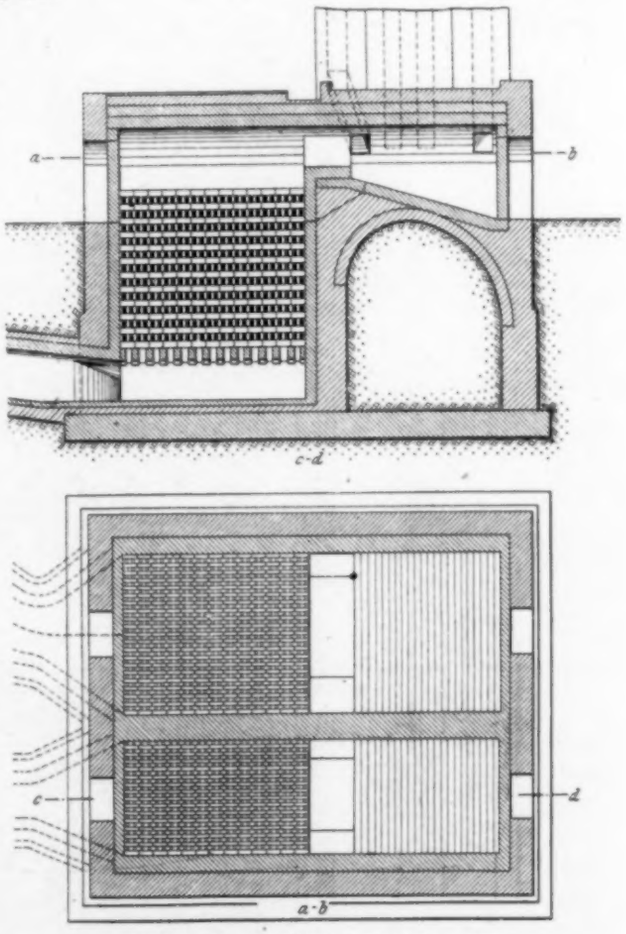


FIG. 15.—ARRANGEMENT OF SLAG POCKETS.

With the latter slags, iron containing carbon can be dephosphorized and all re-phosphorization prevented.

Analyses of a basic steel heat made at Tsaritsin to test this principle showed:

	C.	Si.	Mn.	P.	S.
Pig iron	4.00	0.90	1.90	1.93	0.070
Metal 5½ hours	0.79	..	0.23	0.010	0.008
Final metal	0.61	0.32	0.64	0.012	0.006
	P^2O^5	SiO^2	CaO	Fe^2O^3	FeO	MnO
Slag	5.98	9.46	43.95	5.56	17.87	8.56

	C.	Si.	Mn.	P.	S.
Pig Iron	4.00	0.90	1.90	1.93	0.070
Metal 5½ Hours	0.79	...	0.23	0.010	0.008
Final Metal	0.61	0.32	0.64	0.012	0.006
	P^2O^5	SiO^2	CaO	Fe^2O^3	FeO	MnO
Slag	5.98	9.46	43.95	5.56	17.87	8.56

Index number, or acidity ratio, of the slag, 0.18; metal very quiet in the molds. Test piece, 16 mm (0.64 in.) diameter and 160 mm (6.4 in.) between marks gave 81.2 kg (117,000 lb.) tensile strength and 12 per cent elongation.

Running Open-Hearth Steel Furnaces with Coke-Oven Gas was interestingly discussed by Oberingenieur Terpitz.

*See further on this point, abstract of paper by Richarme. [Translator.]

of the Hubertushütte in Upper Silesia. In the new Georgmarienhütte near Ornabruck blast-furnace gas is used for heating a mixer, but attempts to use it for the steel furnaces have so far failed because of the low heating power of the gas Herr Terpitz is personally convinced that in a few years purified blast-furnace gas will be successfully used in the steel furnaces, particularly if it can be enriched a little.

Coke-oven gas is on the contrary richer than producer gas. A plant of 90 Otto-Hofmann coke ovens is used at the Hubertushütte, which treat 320 tons of coal in 24 hours and have 60,000 cu. m to 70,000 cu. m (2,000,000 cu. ft. to 2,500,000 cu. ft.) of excess gas daily, of a calorific value of 3300 Calories per cubic meter. This gas has been used since 1907 for various heating purposes, and in the steel furnaces as far as it would reach. The gas contains: CO^2, 6.5 per cent; CO, 10.6; N^2, 38.7; N^2, 24.8; O^2, 1.0; CH^4, 16.4; C^nH^{2n+2}, 2.0 per cent. There was enough gas to displace 53.15 per cent of the coal used in the producers for running the furnaces.

The high hydrogen in these gases has had absolutely no injurious effect on the steel; the injurious effects usually ascribed to hydrogen are effects caused by too high a temperature in the furnace, and not to the presence of high hydrogen in the gas used.

Since the coke-oven gases are free from ammonia compounds they should give less trouble from nitrogen in the steel than ordinary producer gas, but Herr Terpitz has no definite conclusions on this head; he thinks the influence of nitrogen in steel still doubtful. The steel takes up no sulphur from the gases and Herr Terpitz takes occcasion to remark that he thinks it never does, under any circumstances, and that the supposed contamination of the steel by impurities in the gases is wholly imaginary.

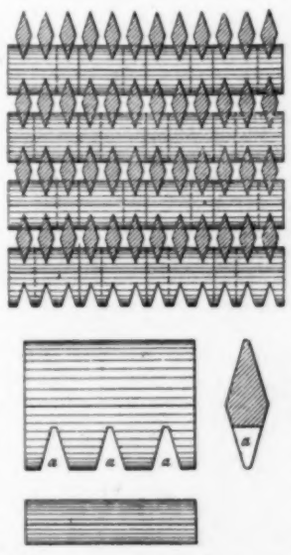


FIG. 16.—CHECKERWORK OF R. SCHERFENBERG.

Concerning imperfect combustion of the hydrogen in the furnace, and its burning in the regenerators, this occurs in their furnaces when fired hard, but not any more with hydrogen than with the CO of the gas; in fact, often less, as analyses show. The higher temperatures of combustion of coke-oven gas have decreased the life of the roof and ports some 8 to 10 per cent. There is no trouble in heating coke-oven gas in the regenerators. If blast-furnace gas and coke-oven gas are both at hand in excess, Herr Terpitz recommends using the former for gas engines and the latter for the open-hearth furnaces. The reviewer would ask why it would not be advantageous for the steel furnaces to *mix* the two gases, since the one being too poor and the other somewhat too rich, a mixture of the two could be easily made with just the desired properties.

Improvements in the Construction of Siemens-Martin Furnaces, by Dipl. Ing. O. FRIEDRICH, of the Julienhütte in Upper Silesia. The rapid development of the open-hearth process, particularly of the pig and ore process, has made great demands on the constructors and workers of these furnaces. The foaming slag rises around the walls even to the ports; fine particles of slag are sometimes carried over by the gases into the regenerators. The voluminous slag also restricts the passage for the heating gases and forces them up against the roof, which then suffers more. The furnaces have also been increased enormously in size, up to steel furnaces for 250 tons (Talbot furnaces) and gas-heated mixers of 1000 tons.

Vertical regenerators are by far preferred to horizontal ones. The air chamber is 1.2 to 1.6 times the size of the gas chamber; 1.4 is recommended as the best proportion, using producer gas. Per ton steel capacity, 1.4 cu. m gas regenerator and 2.0 cu. m. air regenerator space is recommended.

Slag chambers under the furnace are displacing the slag-

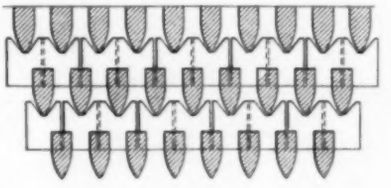


FIG. 17.—CHECKERWORK OF STELLAWERKE.

pockets in the regenerators (Fig. 15). The gases must turn 90° to enter these slag chambers, and this is highly favorable for causing the slag to deposit. The bottom of these chambers is sometimes made removable, on a wagon, so that the fluid slag can be taken away and another wagon inserted very quickly. With the regenerators thus protected, the checker work will stand 2000 to 2500 heats, even on the pig and ore process.

Several of the new forms of checker work in use in Germany are shown in Figs. 16 to 21. Several new forms of gas and air ports are shown in Figs. 22 to 24.

The hearth is usually made 2 to 2½ times as long as broad, in small furnaces; the largest hearths are 4 m (13 ft.) wide and 11 m (36 ft.) long. The bath is seldom over 0.5 m (20 in.) deep. Dolomite is usually used, spread thin upon a background of magnesite bricks. Walls and roof are of silica bricks, except that a lining of magnesite bricks is sometimes run some distance up the walls to protect them from slag in the pig and ore process.

The Present Status of Electric Steel Processes, by Dr. B. NEUMANN, of Darmstadt, was a review of present systems and their results. At the beginning of 1910 there were 67 electric steel furnaces in operation and 40 building. In 1909 17,773 tons of electric steel was made in Germany, mostly fine steel; but the electric furnace is also making "tonnage" steel, and furnaces of 25-tons capacity are quite possible to build. Without going into a detailed description, the following are some of the latest improvements in electric steel practice:

Héroult Furnace.—The author gave an abstract of the article on the South Chicago plant in the April issue of this journal, page 179.

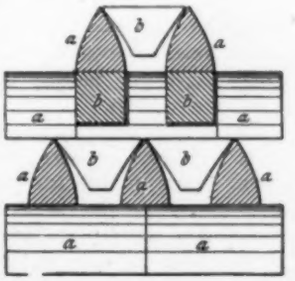


FIG. 18.—CHECKERWORK BRICK OF R. SCHERFENBERG.

Stassano Furnace.—The hearth and side walls are now stamped of burnt dolomite, only the roof is made of magnesite bricks. The roof lasts four to six weeks; the hearth lasts three to four weeks, producing 70 to 100 tons of steel. These figures are for making four heats in 12 hours, and not running nights. When melting scrap and producing steel similar to soft Swedish iron the power used is 750 kw-hours per ton. The bath is deoxidized by silicon and manganese. The rotating furnace has been replaced by a simple tilting construction, which is much cheaper and answers generally quite as well.

Girod Furnace.—The 12-ton furnaces at Ugine are being now charged with 15 to 16 tons of scrap, and 14½ tons are

tapped from them with much better power efficiency.

Chaplet Furnace.—This is used at Allevard; it runs with the same manner of heating and course of the current as the Girod furnace.

Keller Furnace.—Has the whole hearth made conducting by

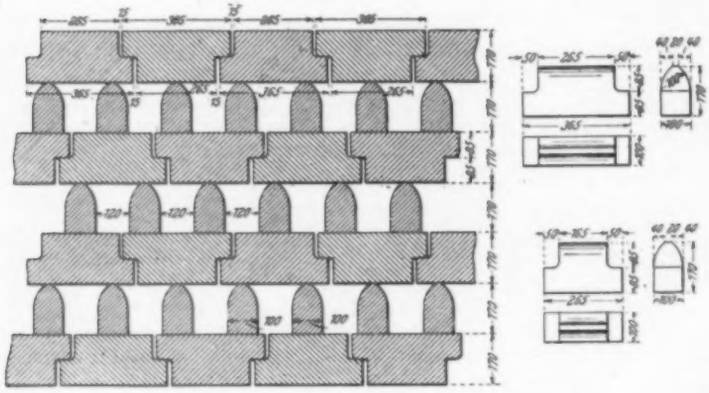


FIG. 19.—CHECKERWORK BRICK OF DIETRICH.

a multitude of small metallic conductors; it has proven itself quite permanent in use. A 3.5-ton furnace of this kind is being installed at Burbach in Germany; this form is preferred to the two electrode Keller furnace.

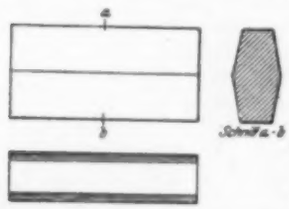
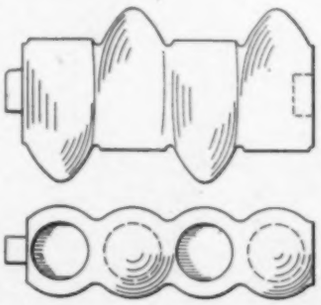


FIG. 20. CHECKERWORK BRICK OF UPPER SILISIAN SCHAMOTTE FABRIK.

Levoz Furnace (Fig. 25).—The furnace body is like a Bessemer converter, but with an electrode passed through the roof and conductors like Keller's hearth built into opposite sides beneath the level of the bath. The whole is mounted on trunnions. This furnace is being tried at a Belgian works.

Nathusius Furnace. — This combines the surface heating of the Héroult furnace with hearth electrodes. Three electrodes of a three-phase system pass through the roof; three steel electrodes, likewise connected with the three-phase system through transformers, are embedded in the hearth, but are covered by the hearth material. As the hearth becomes warm it becomes conducting, and a current of high voltage and small amperage can be sent between these three bottom electrodes, and so the hearth heated to any desired degree (Fig. 26). T represents the secondary of a step-down transformer, T_1 that of an auxiliary transformer for heating from the bottom electrodes.



The bottom electrodes can also be then connected in series with the upper electrodes, the furnace then working like a Girod furnace except that the hearth is highly heated by the passage of the current through it. The bath is kept in active motion by the magnetic field created by the current. If the bath is desired to be left standing, the three upper electrodes can be lifted up and the heating left to the bottom electrodes only.

A 5-ton furnace on this system has been working satisfactorily for a year at the Friedenshütte in Upper Silesia (Fig. 27). The upper electrodes receive 550 kva at 100 volts; the lower set, 150 kva. The furnace is 2.73 m (9 ft.) in diameter outside, and 2.17 m (7 ft.) inside, with two doors and a pouring spout. The electrodes are 2 m long by 0.25 m square, and are suspended from beams quite independent of the furnace, so that the latter stands quite free. The roof is of high-alumina firebricks; 5 to 5½ tons of fluid basic Bessemer steel is treated 3½ to 4 hours, using 300 kw-hours per ton of steel. The product is dead soft steel, with 0.002 to 0.017 per cent phosphorus, 0.005 to 0.020 sulphur, and 0.05 to 0.06 per cent carbon.

Kjellin Furnace.—This simple induction furnace has been built up to 8.5 tons capacity. It is not suited for refining purposes, only for melting, and alloying of special steels.

Röchling-Rodenhauser Furnaces.—These are now working up to 8 tons capacity, and a 16-ton furnace is contemplated. This was the first furnace to make electric steel for rails and railway material. The furnaces are started by laying iron rings in the gutters, and pressing the scrap tightly about them; the latter sinters and soon takes part in conducting the current. Molded dolomite bricks are now used to shut off the slag from the central compartment or reservoir, and so keep the iron in the induction rings clean from slag. The bottom of this central reservoir has also been provided with elevations which contract the section through which the electric current flows and produce locally high heating.

When standing over Sunday, it was usual to keep enough current on the furnaces to keep the contents melted; now the furnace is charged, the doors luted tight and the current shut off entirely. After 30 hours' standing the current is put on again, and in 4 to 6 hours the furnace is at running heat.

The secondary current which is sent through pole plates buried in the hearth material is, when starting, run by a higher voltage, in order more quickly to warm up the hearth and make it conducting; as soon as this is attained, the lower voltage is switched on. When cold scrap is being worked, as much as 2 or 3 tons are charged at once into the 8-ton furnace, the doors closed, and the whole melted down at once without opening the furnace. This has been found to take less energy than the continual feeding in of the scrap in small quantities.

The Frick Furnace (Figs. 28 and 29).—The Fried. Krupp Company has put up a 10-ton furnace of this pattern, which is usually run with a 8½-ton charge, of which 6½ tons are tapped. The heat lasts 6½ hours. It is used only for melting down high quality material to fine steel; the material being charged ¼ ton at a time for 4½ hours, and the last two hours

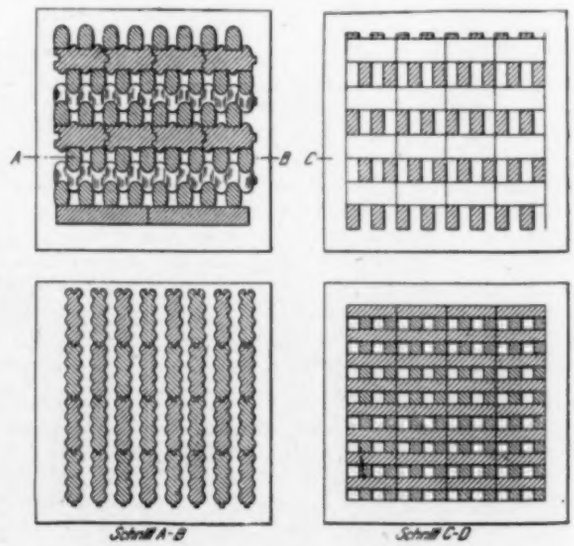


FIG. 21.—CHECKERWORK OF KNOBLAUCH.

being used for heating up to a dead melt. It is an induction furnace which differs from the Kjellin only in the arrangement of the primary windings, which are flat and arranged over and under the melting ring. The outer diameter is 4.4 m

(14 ft.); ring, 2.6 m (9 ft.); breadth of bath, 0.4 m (16 in.).

Primary current, 5000 volts and 265 amp; frequency, 5 per second; power-factor, 0.528; power, 590 kw; calculated amperes in the bath, 52,000; calculated resistance of bath, 0.0002 ohm. Power used, 617 kw-hours per ton of steel. Of the 590 kw, 187 is used to keep the furnace up to heat (to supply radiation and conduction) and 403 for melting and heating purposes; therefore, electrothermal efficiency $\frac{403}{590} = 68$ per cent.

The level of metal in the bath is displaced only 5° from the horizontal, while in the Kjellin furnace it sometimes reaches 24°; the metal in the induction rings rolls actively, which gives

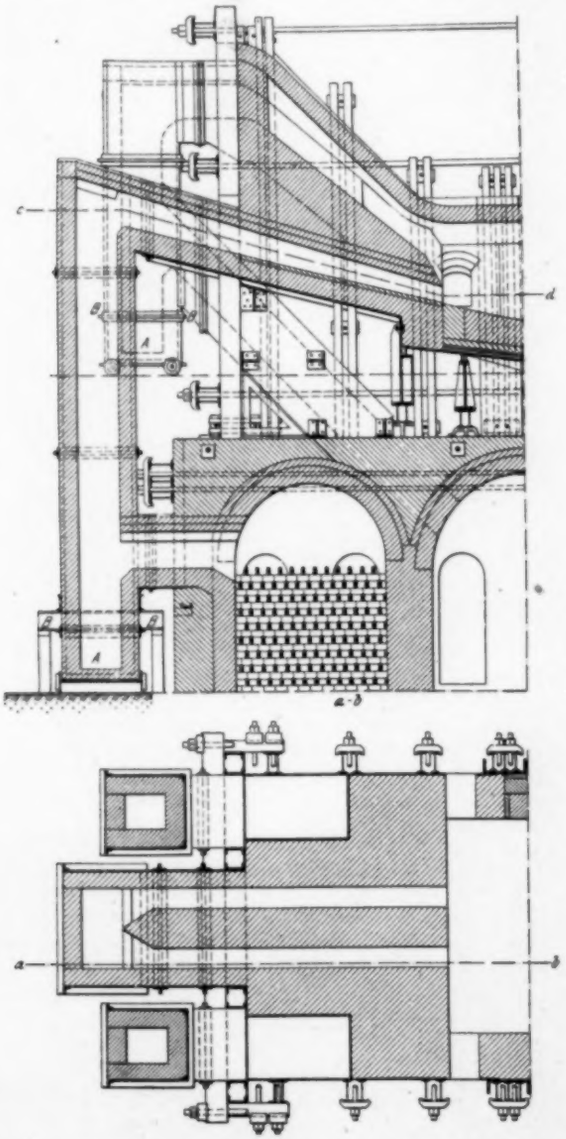


FIG. 22.—ARRANGEMENT OF GAS AND AIR PORTS OF REICHPIETCH.

a good mixture, but makes it difficult to keep the bath covered with slag. In eight weeks this furnace made 180 charges, yielding 1150 tons of fine steel. The campaign was stopped because of injury to the roof; the hearth was uninjured.

Analyses of the steel made show the following properties:

C	Si	Mn	P	S	Tensile strength Kg. sq. cm.	Lb. sq. in.	Elongation. %	Contraction. %
0.38	0.23	0.30	0.020	0.024	50.4	72,000	27.5	65
0.77	0.15	0.40	0.013	0.015	83.7	119,000	16.6	46

The Hiorth Furnace (Figs. 30 and 31).—This has been running in Norway, at Helleren on the Jössingfjord, since December, 1909. It has two rings, like the Röchling-Rodenhauser furnace, with a hearth in the middle. The primary windings are divided into four, which are arranged in flat form above and below (embedded in) the two smelting rings. This results in the maximum electrical transformation efficiency and the smallest dispersion losses. The lower embedded windings are cooled by water or air; the upper ones, in the air, cool themselves, and are movable so that they can be lifted up out of the way. By this arrangement, the smelting ring can be enlarged to any extent desired.

The magnet frame is fixed; only the furnace body with the lower windings is mounted for tilting. The furnace holds 5 to 6 tons; the power-factor is given as 0.7 to 0.8, and the

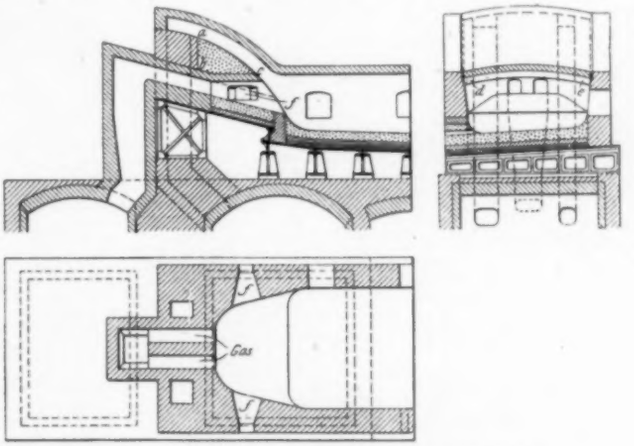


FIG. 23.—ARRANGEMENT OF PORTS BY DIETRICH.

furnace is said to work well, both electrically and metallurgically. The process is only the melting together of pig iron and scrap. Two analyses and tests made of the steel in Stockholm show:

	C.	Si.	Mn.	P.	S.
(1)	0.56	0.14	0.61	0.031	0.01
(2)	1.00	0.16	0.56	0.026	0.015

	Elastic Limit. Kg. per sq. cm.	Lbs. sq. in.	Tensile Strength. Kg. per sq. mm.	Lbs. per sq. in.	Elongation. %	C'traction. %
(1)	38.5	55,000	70.5	100,700	12.3	31.7
(2)	46.4	66,300	97.6	139,400	2.3	2.3

Ischewsky Furnace (Fig. 34).—This is neither an arc nor an induction furnace. It consists of a rotating cylinder with radial bricks *a* between which pass iron electrodes *b*. A 1-ton furnace is in operation, making two turns per minute. The current enters by the brushes *A* and leaves by the brushes *B*, which connect with brass springs *c*, *c*, making connection with *b*, *b*. The current passes partly through the bath and partly through the material of the roof. If no metal is present, the current passes through the copper strip *e*, *e* placed beneath the furnace.

The interior of the furnace is first heated by coke, and some soda put in, to make a thin conducting inner layer. The 1-ton furnace operating in the Urals has 36 iron electrodes (1.1 m x 0.3 m); the inside space is 0.75 m (2.5 ft.) diameter and 1.25 m (4 ft.) long; the walls are 0.3 m (1 ft.) thick. It is lined with acid refractory, and the current used is 600 amp at 550 volts, taken in by four brushes.

In conclusion, Professor Neuman says concerning the development of the electrical manufacture of steel:

"The first stage of the development was in the production of high-priced material, making sharp competition with the costly crucible process. Since the electric furnace furnishes much larger quantities of uniform material, it is not dependent on the purity of the material charged, it works cheaper and deoxidizes the metal at least as well as the crucible wall, it enables the adjusting of the carbon, silicon, and manganese much more accurately, and additions can be made such as it is impossible to make in the crucible. Further, high-quality

material can be made by refining scrap and other impure material, which is a field in which the crucible cannot compete.

"For some time, however, the second stage of the development of electric steel has begun, viz.: the use of the electric

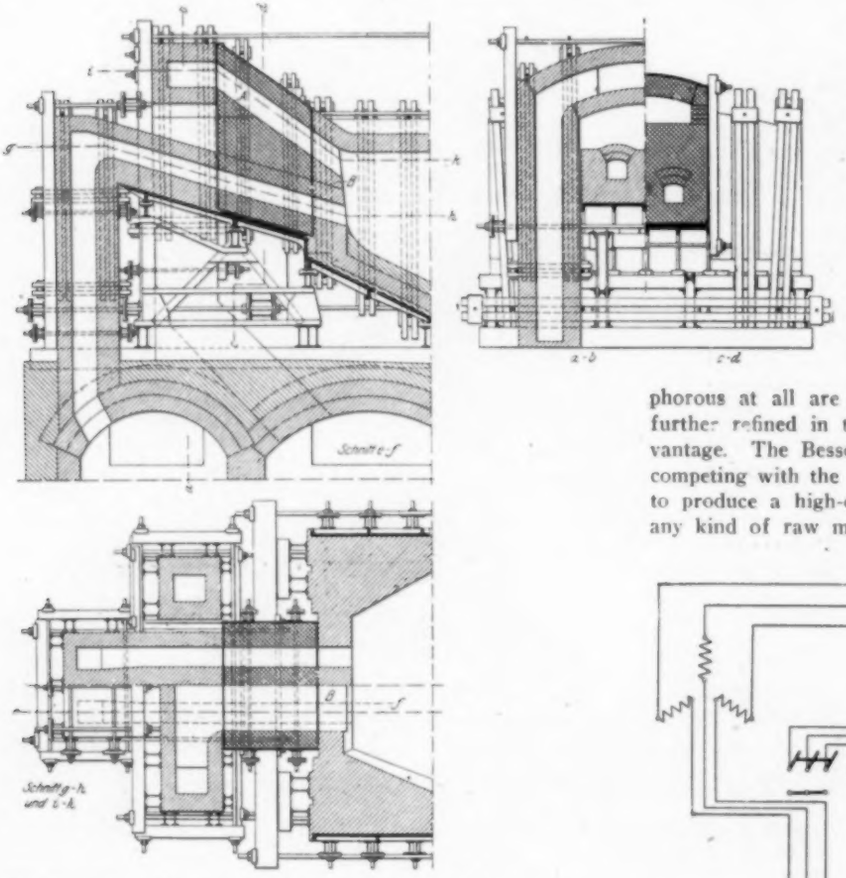


FIG. 24. ARRANGEMENT OF PORTS BY FRIEDRICH

furnace for producing a mean quality of steel to fill in the gap between ordinary steel and crucible quality steel. The electric furnace can be used in combination with the previous steel refining processes to still further refine their product at quite small cost; there is thus produced a nearly sulphur-and-phosphorus-free product, free from segregation, and of great homogeneity, which is particularly suited for high-class rails, tires, axles and springs.

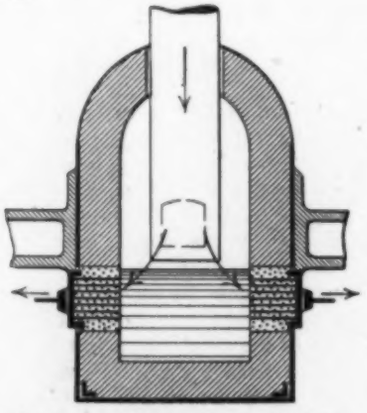


FIG. 25.—LEVOZ STEEL FURNACE.

"According to experience the present used steel refining processes were not used at first because of the cheapness of their product, but because of its superior quality; the electric steel refining will follow the same course. When once a better quality of steel can be furnished for a slightly higher price, this quality is soon used for other and more extensive applications. The electric furnace is well on the way to produce steel in enormous quantities; in the near future it will become a common auxiliary in steel works to supplement the ordinary refining processes, whose entire product will be thus further refined.

"While high-quality steel is mostly made from cold charges, the production of a middle quality in large quantity will require the transfer of fluid metal from converters or open-hearth furnaces, since only in this way is it possible to use the minimum quantity of electrical energy. Since desulphurization in these ordinary furnaces is always incomplete, the advantage of an electrical refining is self-evident. In acid-lined open-hearth or Bessemer furnaces no sulphur or phosphorous at all are eliminated, and the metal from these is further refined in the electric furnace with the greatest advantage. The Bessemer converter may thus find new life in competing with the open-hearth process. It is always possible to produce a high-quality steel in the electric furnace from any kind of raw material, and this will probably make Germany independent of importation of special Swedish raw material, a national economy which is very important.

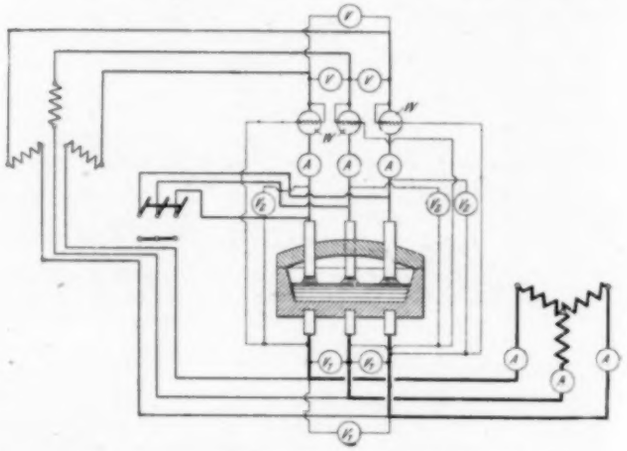


FIG. 26.—CONNECTION OF NATHUSIUS FURNACE.

"A large quantity of electric steel is being made up into steel castings. The electric furnace is a convenient melting apparatus in quite small foundries which are not large enough to run a Bessemer or a small open-hearth furnace. It will also be useful in malleable casting foundries. Why could not fluid cast iron be brought directly from the blast furnace of a mixer or a cupola, put into an electric furnace, and there kept hot and receive necessary alloys until its composition was just what was desired, and then cast at just the proper temperature? This keeping hot need only require 50 kw per ton of metal, so that the certainty of obtaining the exact composition required and of casting at the right temperature would in many cases more than repay the cost of the electrical heating.

"A recent unique application of the electric furnace can be here mentioned. The Eicher-Hütten-Verein, Le Gallais Metz & Cie., at Dommeldingen, convert pig iron directly into steel in a gas-heated, 20-ton tilting Wellman furnace and three 4-ton Röchling-Rodenhauser induction furnaces (Fig. 33). The Wellman furnace acts as an oxidizing mixer. The iron in the Wellman furnace is oxidized down to about 0.26 carbon, 0.082 phosphorus and 0.03 sulphur. This is drawn into the electric furnaces and there refined to the desired quality, using 350-390 kw to a 3.5-ton furnace, in about two hours to steel for

castings and 3 to 3½ hours for best machinery steel. Some analyses are:

	C	Si	Mn	P	S
Pig Iron.....	3.50	0.60	1.20	1.80	0.12
Mixture in Wellman....	1.62	..	0.66	0.46	0.035
Tappings from Wellman.	0.26	..	0.30	0.08	0.030
Steel Castings.....	0.35 to 0.40	0.25	0.50	0.020	0.010
Hard fine steel	0.85	0.23	0.26	0.011	0.009
Chrome steel Cr. 0.87	1.17	0.19	0.16	0.014	0.009

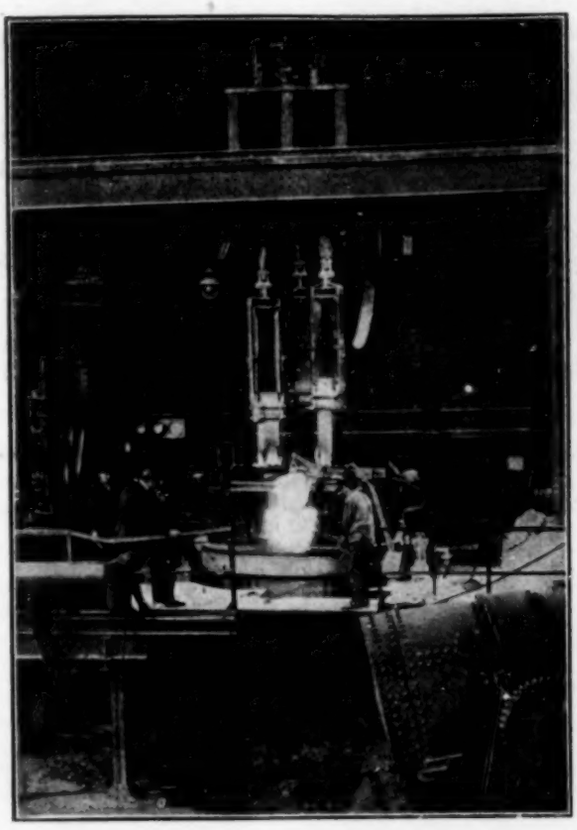


FIG. 27.—NATHUSIUS FURNACE AT FRIEDNHÜTTE.

"The above review gives an idea of the extension of the field of the electric furnace. With such a young branch of industry it is naturally not to be predicted what further possible applications may develop. So much, however, is sure, that the introduction of the electric-steel furnace has opened up a new era of refining to high-quality steel, at a moderate cost."

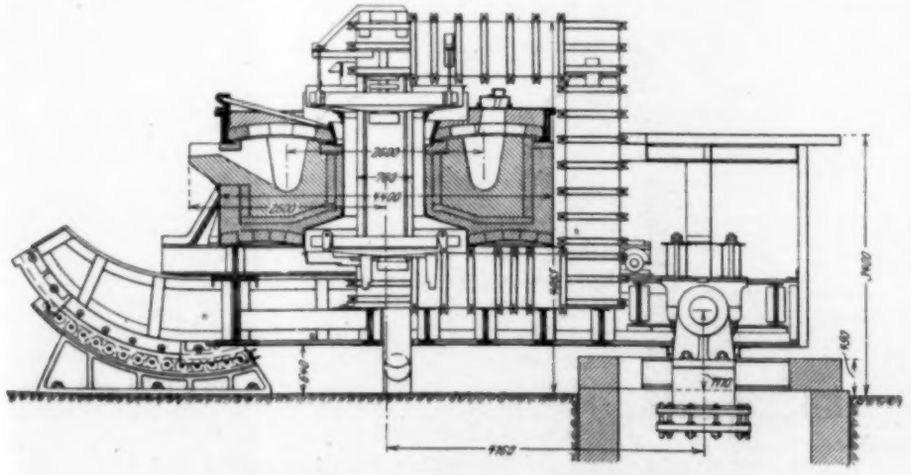


FIG. 28.—LONGITUDINAL SECTION OF FRICK FURNACE.

Acid and Basic Electric Furnaces were discussed by Director O. THALLNER, of Remscheid, from a new standpoint. He tried to find an explanation for the different effects of the acid and basic linings on the steel. All charges melted at a low temperature on basic hearths give steel of poor structure and low physical qualities; but charges which are smelted on a basic hearth with the temperature gradually (not suddenly) raised to a high point, give good steels. Acid hearths, on the contrary, give good steels whether the heat is high or low, and whether the temperature is raised fast or slowly. Director Thallner thinks that the well-known reaction of the carbon in the steel on the silica hearth "supports very strongly the breaking up of the atomic grouping of the steel molecules, and so produces good quality steel." How it does it, or why it should do it, he does not say; and, besides, the assumed atomic groupings are highly hypothetical, resting upon no firmer basis than the following piece of argumentative reasoning: "If changes in the structure of steel are produced by thermal treatment at temperatures below the melting point, one cannot deny similar results from heat treatment above the melting point." One might feel free to deny it until it was proved, or facts adduced to make it fairly probable. The reasons why the basic lining gives poorer results at low temperatures are not given. Altogether, the paper seems merely a piece of metallurgical speculation; the facts adduced are probably correct, and important; the attempted explanations seem far-fetched and improbable.

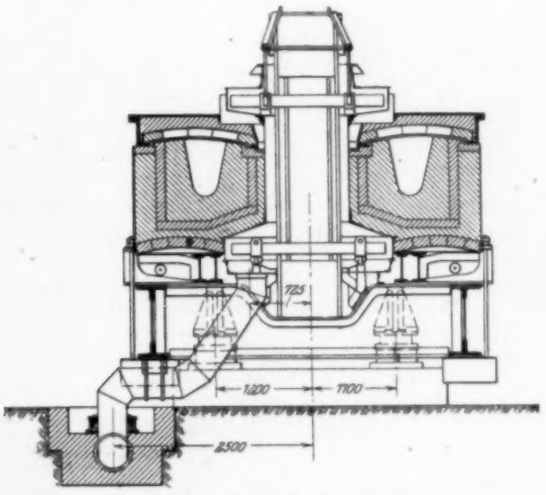


FIG. 29.—VERTICAL SECTION OF FRICK FURNACE.

Current and Voltage in Electric Furnaces, by Dr. Ing. W. CONRAD, of Vienna, records a measurement of the relative amounts of energy developed in the bath of a Héroult furnace and in the arcs above the bath. With the electrodes short-circuited in the metal the resistance of the bath was 0.00027 ohm, there being present 185 liters of melted steel. From this it is calculated that only 0.73 per cent of the energy of the current is expended in overcoming the resistance of the bath. The author attempts to calculate the resistivity of the steel from these data, but the calculation is a miss, because of the irregular shape of the bath. If he were to determine the resistance for a similar shaped bath of a liquid of known resistivity, the calculation could be made.

Notes on Electric Steel, by G. ARNOU, of Paris, deals with

the general quality of the steels made in the Savoy. P. Girod's large works at Ugine has made from June, 1909, to May 1, 1910, 8000 tons of ingots and 850 tons of castings. The Hauts-Fourneaux et Forges d'Allevard have made steel since 1904 in Chaplet furnaces, possessing two furnaces of 3 tons, two of

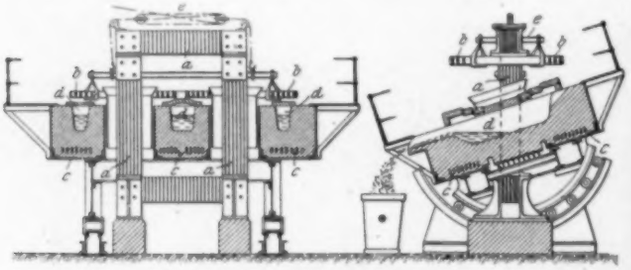


FIG. 30.—SECTIONS OF HIORTH FURNACE.

5 tons, and a larger furnace under construction. They have sold altogether several thousand tons of electric steel.

The absence of gas and oxides and its remarkable homogeneity result practically in two very valuable qualities: a high degree of malleability when hot, and a very strong resistance

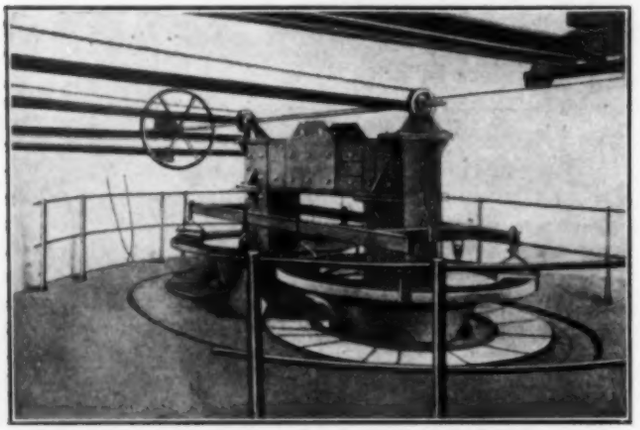


FIG. 31.—HIORTH FURNACE.

to shock. For an equal content of carbon, manganese, tungsten, etc., and given exactly similar treatment, the electric-steel shows 5 per cent more initial magnetism than the non-electric.

Mr. Arnou then gives some remarks which explain the differ-

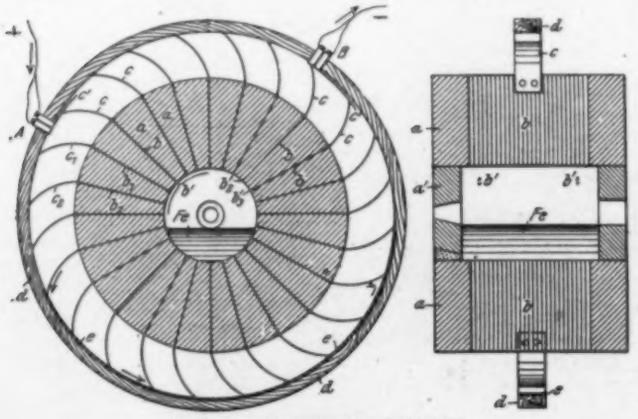


FIG. 32.—ISCHEWSKY FURNACE.

ence between acid and basic practice much more satisfactorily than the paper of Director Thallner. The open-hearth metal always contains dissolved FeO, which on addition of Mn becomes mostly MnO. In contact with a richly basic slag, these oxides are only slowly washed out of the metal; siliceous slags, on the contrary, quickly absorb these dissolved or suspended oxides from the bath. In the electric furnace, with acid lining, these oxides are rapidly removed from the metal, and good steel results; in basic lined furnaces they only come out slowly, by slow heating to a high temperature, and letting stand. In

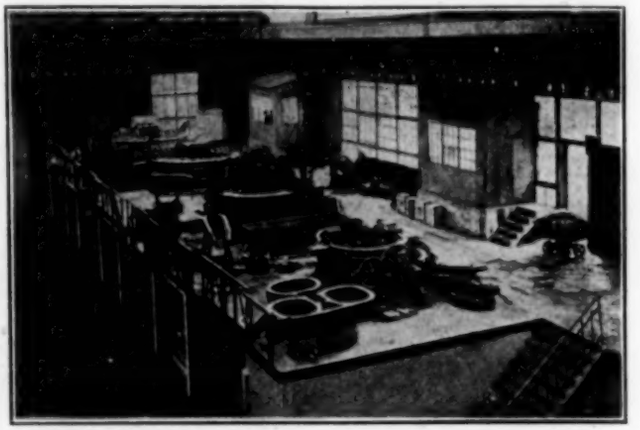


FIG. 33.—ELECTRIC STEEL FURNACES IN DOMMELDINGEN.

fact, the microscopic observation of these steels shows when the suspended basic oxides (FeO, MnO) have been removed and when they have been left in; the quality of the steel always corresponds.

The electric steel castings, such as those made by the Usine Girod, are of such good quality that they can in many places replace forged steel articles, so that a vast field is here opened up for the electric-steel industry.

On account of limitations of space some further abstracts had to be reserved for publication in later issues.

Sinking Heads on Castings.

At the recent meeting of the Association of Water Engineers, at York, Mr. T. Kennedy read a paper on the "Effect of a Sinking Head on Large Castings." He described the results obtained in casting some iron columns 32 ft. long and 12 tons in weight, to which sinking heads of 2 ft. 10½ in. long and about 2 tons in weight were given. After detachment, the head was broken, and the fracture showed that the head—which had a considerably greater area than the column—had perfectly maintained the requisite supply as contraction went on in the molten metal of the column. Further, large pockets were formed, and in these there were masses of crystals of pure iron in interlaced groups, while the other materials combined with some of the iron and formed a matrix filling the interstices. The strength of the casting depended on such interlaced groups which were found in identically the same form, but of much smaller size, in the body of the casting where they were completely interlaced and homogeneous as a result of the pressure exercised by the more slowly cooling sinking head. The worst impurities in cast iron as it came from the cupola were globules of slag, free gases and occluded gases. The last named were absorbed physically in the metal, and were, to a very great extent, liberated during the process of solidification. During the pouring more gases were given off from the sand, and for the greater part escaped through the cores and the box joints. The impurities had a lower specific gravity than the iron, and rose to the top if there were a free passage in the mold; if there were not, they were retained, and caused porosity and weakness in the casting. If a long column were cast on a slope instead of vertically the millions of crystals which formed round the mold at right angles to the cooling surfaces greatly hindered the separation of impurities; but if the column were cast horizontally the slag and gases could not get away and would collect in the top side of the casting.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Failure of Light Engineering Alloys.

Mr. EDWARD F. LAW read a paper before the Faraday Society on June 21 on the failure of light engineering alloys, particularly aluminium alloys, in which he contended that the cause of the disintegration was an allotropic change, and suggested that this occurred in the aluminium, and was not dependent on the presence of other metals, although they facilitated its production. This view was supported by the fact that some samples of "pure" aluminium gave a well-defined break in the cooling curve at 632° C. Other specimens, however, did not exhibit this feature. Aluminium expanded in solidifying and, as Professor Turner had shown in a recent paper read at the Institute of Metals, its behavior during solidification and cooling was similar to that of antimony and zinc.

In opening the discussion the chairman, Dr. F. M. Perken, remarked that other metals besides aluminium broke down in the way described by the author—for instance, chromium, manganese and calcium.

Dr. R. Seligman said some of the questions raised had occurred to him years ago, when he noticed the want of cohesion in aluminium below its melting point. The phenomenon occurred in aluminium of the highest purity; and, therefore, he questioned its being caused by impurities. If an allotropic form of aluminium existed it must be very weak and completely useless. Four years ago he sent some specimens of aluminium to Dr. Harker, who might throw further light on the behavior of this metal.

Dr. J. A. Harker said he had determined the freezing points of the samples of pure aluminium he received from Dr. Seligman, and had looked for the lower point at which the metal became friable. He found a slight indication of a kink in the cooling curve at about 630° C. (the melting point was 657° C.); but otherwise the results were not very satisfactory nor conclusive.

Dr. W. Rosenhain considered that the failure of aluminium alloys was not so great as Mr. Law's opening remarks indicated, or they would not be used in ships in preference to high-tensile steel. With regard to disintegration, he had watched this going on in aluminium-manganese alloys under the microscope. The process resembled miniature volcanic eruptions, fragments of crystals being ejected from the mass. These polymorphic changes were not restricted to aluminium alloys, nor even to metals; they occurred in certain silicates of calcium. He had measured the freezing points of more than 50 specimens of aluminium; but absence of sharpness in the freeze occurred in only one or two instances, and in these the metal must have been contaminated. He had no doubt that allotropic changes in metals were always dependent on the presence of other metals. Dr. Rosenhain was understood to attribute the second break in the curve to a eutectic.

Mr. W. H. Merrett remarked that some years ago he had experimented with a large number of aluminium alloys, and he had invariably found that where the initial freezing points were identical there was afterward a falling off amounting to several degrees. The addition of 5 per cent of bismuth greatly improved the quality of aluminium for casting and turning.

Mr. E. Ristori suggested observations of the effect of putting mechanical work into spontaneously decomposable alloys as soon as they were cast.

Dr. T. M. Lowry observed that most metals when alloyed with iron increase its hardness and retentiveness but decrease its permeability. It was remarkable that in the case of aluminium and silicon the reverse occurred. Could any comparison be traced between the two series of elements with regard to effect on the transition temperatures of iron?

Mr. Law, in reply, said he thought that failure was quite as great as he had stated. [Dr. Seligman signified acquiescence.] Dr. Rosenhain had brought forward several arguments in support of his (the author's) theory. They might not have been intended to support it; but if carefully examined they would be found to do so. He (Mr. Law) had not stated or implied that disintegration did not occur with other metals; aluminium was not unique in that respect. He was of opinion that there was not any eutectic at the second break in the curve. There were three distinct evolutions of heat, and only two could be accounted for with a eutectic when only two metals were present. Fifty freezing points and coolings could always be done without finding the second break when one wanted to obtain it; but when it was not wanted it appeared. The publication of unsatisfactory and inconclusive results of researches might often prove very valuable.

The Electro-Metallurgy of Ferro-Alloys and Steel.

The discussion on M. PAUL GIROD's paper read before the Faraday Society on June 21 was opened by Dr. F. M. Perkin, who remarked that progress in electric steel refining in England appeared to be slow, but possibly more had been done than was generally known. The cost of power for manufacturing special steels was not so all-important as it was sometimes thought to be.

Dr. W. Rosenhain said that if the best tool steel could be made in the electric furnace from scrap, a revolution in the Sheffield industry must follow. The Sheffield manufacturers fully appreciated the situation, and were experimenting carefully and systematically.

Mr. F. W. Harbord remarked that M. Girod merely claimed advantages in respect of form of the furnace and arrangement of the electrodes. Equally good results could be obtained with any furnace if facilities existed for removal of the slag charged with impurities. The problem of the electric steel furnace was to devise the best mechanical and electrical appliances for the two essentials of refining, viz., oxidation and reduction. Several electric steel furnaces were at work in England. There was not any doubt that steel could be made from scrap in the electric furnace equal to the best crucible steel.

Mr. E. Ristori thought the paper exaggerated the author's claim for priority in respect of electric steel refining, although possibly not in the case of ferro-alloys. He did not consider that the modifications introduced were improvements.

British Aluminium Reconstruction.

At the statutory meeting, on June 29, of the British Aluminium Company (Ltd.), the chairman, Mr. A. W. Tait, said that the company's reconstruction had been most successfully carried out under the terms of the circular of February last. The whole of the shares were issued and allotted, and they had also issued £800,000 of 5 per cent prior lien debentures at the price of £97 per cent, so that the new company would have a total cash sum of £974,435 when the whole of the calls were paid up. The capital of the new company was a total of £2,420,581, made up as follows: £800,000 of 5 per cent debentures, £720,398 of debenture stock, £299,570 of non-cumulative preference shares of £1 each, and £600,613 in ordinary shares. It was very gratifying that only a very small percentage of the shareholders did not take up the interest to which they were entitled in the new company. The company would be able to find the funds for the completion of the Loch Leven Works, and for all that was necessary in regard to other works, and have money left in hand. It was impossible at the statutory meeting of the company to accurately forecast the profits, but the works of the company were in good order, and there was every reason to expect that a considerably larger tonnage of aluminium would be produced during the current year. If the present demand for the metal continued to expand they should be able to get rid of the whole of the output from their various works at a price which would yield a reasonable profit. Of course, no one could ex-

pect that the aluminium industry would recover immediately from the bad time of 1908, and the greater part of 1909, but the market was undoubtedly broadening and prices had already shown some small recovery. There was every reason to look forward with confidence to the future of the reconstructed company.

Market Prices.

JUNE, 1910.

Copper remained steady until the middle of the month, then falling from £56 10 0 to £54 11 0. Since recovered slightly.

Tin has shown a downward tendency, dropping from £149 to £147.

English lead has remained steady at or in the neighborhood of £13.

Iron, Hematite has fallen off from 66/2 to 65/-.

Scotch Pig has fallen steadily from 55/10½ to 55/-.

Cleveland Warrants have dropped from 49/10½ to 49/1; slight recovery later.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, per ton	30	0	0
Borax, best refined crystal, per ton	16	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	11	10	0
Sal-ammoniac, lump, first delivered U. K., per ton	42	0	0
Copper sulphate, per ton	17	10	0
Camphor, 1 oz. tablets		1	7
Caustic soda, 48 per cent ordinary, per ton	5	10	0
Bleaching powder, 35 per cent, per ton	4	5	0
Shellac, standard T. N. orange spot, per cwt	3	13	0
Sulphur, recovered, per ton	5	0	0
Carbolic acid, liquid 98/99 per cent, per gal		1	0
Naphtha solvent, 90 per cent at 160° C., per gal		2	4
Platinum, per ounce, nominal	6	0	0

Rubber, while touching 9/2 on the 8th, ultimately rose to 10/-, and subsequently again fell to the neighborhood of 9/4.

London, June, 1910.

SYNOPSIS OF METALLURGICAL AND CHEMICAL LITERATURE.

Iron and Steel.

Gases in Commercial Iron.—P. GOERENS, of Aachen, presented a paper on this subject at the International Metallurgical Congress at Düsseldorf. This is reprinted in a recent issue of *Metallurgie*. The first requirement for producing sound steel-castings is the absence of gas in the metal; in steel for rolling and forging the presence of considerable gases is undesirable, since the waste from cropping the ingots will be larger and the blow-holes will roll out to scales on the finished steel. After reviewing the history of the subject, the author describes his tests. Samples of steel were taken at various stages of the process, cast into ingots, and these investigated for dissolved and mechanically-included gases when heated to temperatures of 850° to 1000° C, in a vacuum. The results are contradictory, leaving it very doubtful whether they really represent the composition of the liquid metal. The results must be taken for simply what they are, viz., gases given off by the metal first cast in the air and then reheated to a bright red heat in a vacuum.

Basic Bessemer Metal, at end of blow:

	1.	2.	3.	4.	5.	6.	7.
Volume of gas per 1 Vol. Metal.	0.88	1.95	1.44	1.5	0.59	2.1	2.2
Percentage of CO^2	20.1	13.8	9.7	6.5	9.5	24.8	9.9
Percentage of CO	42.1	45.1	51.8	76.6	64.1	46.5	68.2
Percentage of H^2	24.2	31.8	25.5	11.3	15.9	17.4	20.1
Percentage of N^2	13.6	9.3	13.0	5.6	10.5	11.3	1.8

Basic Bessemer Metal, after Fe-Mn. addition:

Volume of gas per Vol. Metal	3.38	4.1	3.3	3.9	4.4	2.7	2.1
Percentage of CO^2	7.75	14.7	8.8	1.5	4.3	5.8	1.1
Percentage of CO	76.3	65.1	69.9	77.2	79.7	73.4	—
Percentage of H^2	6.0	12.2	11.0	4.5	3.5	7.8	22.2
Percentage of N^2	9.95	8.0	10.3	16.8	12.5	13.0	—

Open-hearth basic steel:

	Before final additions.	After final additions.	From last ingot cast
Volume of gas per 1 Vol. metal	1.7	1.8	3.3
Percentage of CO^2	5.0	2.6	4.2
Percentage of CO	78.5	83.2	81.7
Percentage of H^2	7.4	4.7	10.2
Percentage of N^2	9.1	9.5	3.9

Electric steel ready for casting:

	After removing first slag.	After removing second slag	Finished steel.
Volume of gas per 1 Vol. metal	1.3	1.3	1.9
Percentage of CO^2	7.5	6.4	4.7
Percentage of CO	80.1	84.8	78.3
Percentage of H^2	7.4	4.7	11.7
Percentage of N^2	5.3	4.1	5.3

Crucible Steel:

	Charge 1.	Charge 2.
Volume of gas per 1 Vol. metal	1.9	1.5
Percentage of CO^2	1.3	0.7
Percentage of CO	69.2	71.6
Percentage of H^2	18.6	7.2
Percentage of N^2	10.9	20.5

Miscellaneous Material:

	Steel rail.	Swedish pig-iron.
Volume of gas per 1 Vol. metal	6.1	8.2
Percentage of CO^2	3.7	11.5
Percentage of CO	72.7	70.5
Percentage of H^2	8.3	11.1
Percentage of N^2	15.3	6.9

A careful study of these tests will show practically no connection between the volume of gas and the nature of the steel. It will show, however, that tests made as these were are of no value whatever in forming conclusions as to the nature and amount of the gases in the *fluid* metal, or of the reactions by which they are to be reduced in amount or prevented from spoiling the solid casting. The author, in trying to draw some conclusions, comes to the contradictory result that deoxidizing processes in general enrich the steel in gas, and therefore damage the steel. The truth is, that the tests are incapable of giving any reliable information on the behavior of the fluid steel in casting.

Gold and Silver.

The "De Wilde" Process of Precipitating Gold and Silver from cyanide solutions was devised about 15 years ago by Prof. P. De Wilde of the University of Brussels. In brief, this process consists in acidulating the cyanide solution with sulphuric acid with the addition of cuprous chloride. A yellowish precipitate is formed which consists of an intimate mixture of aurous cyanide and cuprous cyanide. Silver is precipitated as chloride by the addition of sodium chloride with the sulphuric acid. In the April, 1910, *Informes y Memorias* of the Mexican Institute of Mines and Metallurgy, Mr. G. WITTEVEEN contributes the results of some experiments performed to determine the practical utility of the scheme as applied to silver solutions. In his description of the process, Prof. De Wilde stated that "the solution (after precipitation) contains all the cyanogen less the small amount combined in the gold-copper precipitate and the very small quantity volatilized as cyanhydric acid." The author of this article believed that this might be true of very weak solutions used for gold ores, but doubted the statement as applied to the strong cyanide solutions used in silver cyanidation.

To test this he took 250 cc. of cyanide solutions of 0.05 per cent to 0.40 per cent KCN acidified each with 0.4 cc. H_2SO_4 to each 0.05 per cent KCN. The solutions were stirred and allowed to stand for two hours, when they were made alkaline with a 6 per cent caustic soda solution. The solutions were then titrated for KCN and the loss found to be almost uniformly 10 per cent. The same experiment repeated, with neutralization by alkali immediately after acidification, showed the loss to be practically nil. The experiments showed that the loss of cyanide depends upon the time elapsing between acidification and neutralization, and since considerable time must elapse in actual practice the author concludes that the process is impracticable owing to loss of cyanide. Further experiments on working cyanide silver solutions containing 0.25 per cent KCN and 225 grams of silver per ton gave such inconstant results that it was impossible to base on them an estimate of the working costs of the process, but they were found to be prohibitive. The conclusion is that the De Wilde

process offers no advantage in the precipitation of silver bearing cyanide solutions.

Cyanide Plant of the Consolidated Mercur Co.—This plant is one of the most interesting in the west on account of the developments in cyanidation which have found their beginning here. Cyanidation on a large scale, the treatment of base ores and the vacuum filtration of slime were first successfully tried at this property. The present system is described at length in the June 18, 1910, issue of the *Eng. & Min. Journ.*, by Mr. ROY H. ALLEN.

Two classes of ore are treated; oxidized ore is treated direct, but the base ore receives a preliminary roast to prepare it for cyanidation. The oxidized ore is crushed and passed over screens having ⅝-in. openings, the undersize passing to the fine ore bins. The oversize is crushed again and passed through trommels fitted with ½-in. screens. The oversize from the trommels is cyanided direct, and the undersize is combined with the fine ore. The latter is mixed with cyanide solution through Mercur mixer-classifiers in which the coarse sand is separated from the fine and slime. The coarse sand is subjected to a second similar operation after which it is conveyed to charging bins, whence it is drawn to be mixed with the coarse ore for leaching. The fine and slime are treated in cone classifiers, the spigot product of which is further classified in Dorr classifiers. The sand from these classifiers is sluiced to leaching vats, while the slime is thickened in Dorr Slime thickeners. The treatment of the slime is complete in these thickeners and the pulp is passed from them to the filters. The Mercur filters are of the stationary vacuum type. There are 104 leaves 7 ft. square, a vacuum of about 20 in. of mercury is used, and under this vacuum it takes about 50 minutes to form a cake ⅝ to ¾ in. thick. Fifty minutes are required for the washing and about 20 minutes for the transfer during the discharging process, thus consuming about two hours in a complete cycle.

The base ore is crushed and screened through 1/6-in. stationary screens and trommels and delivered to five straight-line roasters, one Holthoff-Wethey, one Brown and three Jackling, in which it is roasted for about six hours during its passage through the furnaces. Leaching is carried on in 18 rectangular vats, each 25 ft. x 50 ft. x 4 ft. 10 in., with a capacity of 225 tons of sand. Four of these tanks are used for fine sand from the Dorr classifiers, twelve are used for coarse oxidized and roasted ore and coarse sand, and two are held in reserve. The fine sand tanks are filled by sluicing, while the others are filled by hand, the ore being trammed over the tanks, dumped and spread. Cyanide solution is run in from the bottom for about eight hours, allowed to stand for four hours, when leaching is commenced. Strong solution is applied for 24 hours and weak solution for 48 hours. Washing requires 12 hours. Zinc dust is used as a precipitant, and 10 Johnson filter presses are required to handle the solutions. For the year ended June 30, 1909, the following data appear: Mill ore assayed $3.58 per ton; tailing $0.88 per ton; extraction 75.4 per cent; consumption of cyanide, 0.66 lb.; lime, 5.4 lb.; caustic soda, 0.05 lb.; zinc, 0.45 lb. per ton of ore treated. The cost of milling 282,269 tons of ore is given as $1.09 per ton. The cost of roasting, $1.05 per ton of ore roasted.

Simplifying the Cyanide Process.—In our issue for February, 1910 (Vol. VIII, No. 2, p. 72) there appeared a description of Mr. A. F. Crosse's system of slime treatment. The main feature of the process consists of the elimination of the filter press, and precipitation of a clear cyanide solution obtained by decantation. The suggestion has been taken up by Mr. BRUNO MIERISCH, who describes in the *Eng. & Min. Journ.* for June 25, 1910, certain proposed simplifications of the cyanide process. The proposal provides a continuous system of treatment in which the handling of the ore is reduced to a minimum.

In flow-sheet *A* (Fig. 1) there are as many "Crosse" tanks as the number of days required for the treatment of the ore, plus three more of identical design. Thus if the treatment requires three days, the number of tanks is six: three for the treatment proper; one which is being filled with ore; one being emptied and the sixth receiving the overflowing solution. Assuming that tanks Nos. 2, 3, 4 and 5 are filled with pulp, No. 1 is filled with the necessary cyanide solution and receives its charge from the mill, and No. 6 is empty. Air is turned on in all tanks except No. 6. Thickened pulp flows into the first tank and displaces a corresponding volume of solution which flows through No. 1 zinc box and into tank No. 2. Here a similar overflow is effected after mixture with the pulp, and so on to No. 4, from which zinc box the solution is pumped into No. 6. The ore in tank No. 5, having given up its gold and silver, is washed with an amount of water lost by evaporation and by that retained in the discharged slime. The resulting dilute solution is returned to the crushing and grinding apparatus. The cycle of operations advances from tank to tank each 24 hours and goes on continuously. In flow-sheet B zinc box precipitation is eliminated and amalgamating pans substituted. Only five tanks are required. As the solution flows into tank No. 5 the thickened pulp is taken out and directed into the amalgamating pans which contain mercury, zinc amalgam and copper. After about a two-hour treatment the gold and silver are precipitated into the amalgam and recovered as usual.

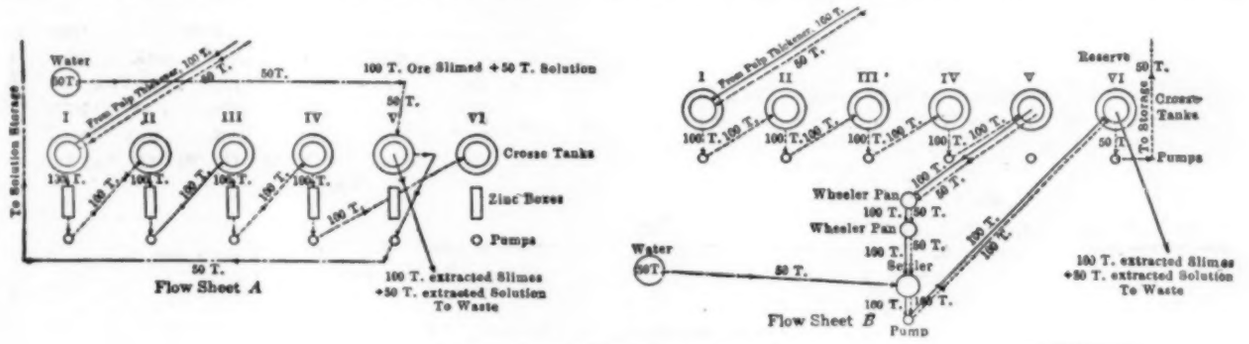


FIG. 1.—SIMPLIFYING CYANIDE PROCESS.

Neither of the systems has been tried in practice, but the author believes that they offer theoretical advantages which merit trial.

Aluminium.

The Aluminium Industry in France is one of growing importance. The principal deposits of bauxite occur in the southern part of France and comprise two varieties, white and red. The former is low in iron, but high in silica, while the latter is higher in iron and low in silica. Between these two varieties are intermediate grades in which the impurities of iron and silica replace each other. In the June 11, 1910, issue of the *Eng. & Min. Journ.*, TONY CALLOT gives some details of the French aluminium industry. The percentage of Al_2O_3 varies from 55 to 70 per cent, while an unusually pure variety yields

78 to 80 per cent Al_2O_3, with almost no iron, 2 to 3 per cent silica and 2 to 3 per cent TiO_2. The best ore is found in the Department du Var, where it is sold under a formal guarantee to contain 60 per cent Al_2O_3 and a maximum of 3 per cent silica.

Two processes are used in France to produce alumina from bauxite. The Bayer process is used at Gardanne, where the ore is attacked under pressure with caustic soda solution, producing sodium aluminate which is separated by filtration. Alumina precipitates from this spontaneously, or the precipitation may be aided by the addition of some prepared alumina. The precipitated alumina is washed and calcined, when it is ready for reduction to aluminium. In the other process which is used at Calypso, the pulverized bauxite is charged in ordinary soda furnaces with soda. The sodium aluminate thus obtained is dissolved in water, filtered from the iron and then treated with CO_2 to precipitate the alumina. The reduction of alumina to aluminium is accomplished by the well-known electrolytic process. A large quantity of electric horsepower is required. About 100,000 hp is used annually.

Copper.

Basic Lined Converters.—The first successful use of basic lining for converting was by Knudsen, who developed his system of pyritic converter smelting at Sulitelma, Norway. Other more or less successful attempts to use basic lining have been made by other metallurgists in this country for twenty years. Redick R. Moore describes the present practice in the United States in the *Eng. & Min. Journ.* for June 25, 1910. The data given are from operations at Garfield, Utah and Perth Amboy, N. J. The steel converter shell is of ¾ in. steel and is 23 ft. long and 10 ft. inside diameter. There are 32 1¼-in. cast-iron tuyeres. A circular throat 40 in. in diameter is placed at one end; other openings are provided for removal of slag and metal, and for the examination of the tuyeres, and for the introduction of an oil burner. The slag opening and the opening for inspection of tuyeres are at the side opposite the latter, and the opening for the oil burner is at the end opposite the throat. These openings are kept luted with clay when not in use. The lining is of 9-inch magnesite brick except at the tuyeres, where 18-in. brick are used. The oval opening inside the lining is about 6 ft. 3 in. high by 7 ft. 3 in. wide by 19 ft. 9 in. long. The brick are set in magnesite powder, except at the tuyeres, where a mixture of magnesia and linseed oil is used.

When seasoned, an initial charge of 30 to 40 tons of matte is poured in through the throat in the usual manner. About 6000 pounds of silicious ore are added and the blast turned on, the pressure being about 5 lb. for a few minutes, being later increased to twice that amount. Blowing is continued for 30 to 45 minutes until the silica is fluxed, when the slag is poured. Another ladle of matte is added and another charge if silicious ore. This is kept up until the converter is filled to within 2 to 4 inches of the tuyeres with white metal, when the charge is blown to copper. The appearance of the flame is not relied upon when blowing leady or arsenical mattes, but test samples are taken from time to time. Slags are usually lower in silica and copper than those from acid lined converters; they are generally made to carry about 27 per cent silica. With leady mattes this can be reduced to 23 per cent.

Foaming is caused in basic converters when the charge is too cold and when an excess of silica has been added. Leady matte is also prone to foam badly, but foaming can be corrected by the quick addition of a ladle of hot matte and increase of the blast. The basic converter also shows greater radiation and conduction heat losses than the acid converter, necessitating working with larger charges or lower grade mattes and higher blast volumes. Also on account of the lower temperature it is necessary to keep men constantly punching tuyeres.

The advantages of the basic lined converter are stated by the author as follows: Cheaper construction, less building space and lighter crane, large vessels may be used as there is no handling of the same, no clay used for binder in relining, less slag produced for further smelting to recover copper, and lower grade mattes can be converted at a less cost per ton of matte and silicious ore than the same concentration can be effected in the blast furnace. The disadvantages are: Loss of fines due to blowing out of the throat, lost time in relining and repairing, extra careful manipulation required to avoid burning out lining and tuyeres and the formation of accretions, and extra tuyere punching necessary. These advantages and disadvantages will determine the use of the basic converter in different localities.

Electrolytic Refining.—An extensive paper on electrolytic copper refining was presented before the Australasian Institute of Mining Engineers, and published in their *Proceedings*, by G. H. Blakemore. The article describes in detail the procedure at Lithgow, New South Wales, where the author was engaged for some years. Efficient insulation, cleanliness and temperature of the electrolyte are discussed as the three secrets of successful electrolytic refining. The making of starting sheets, the parting of the Dore bullion, filtration and purification of electrolyte, effect of slime suspended in electrolyte, and the methods of assay used are all discussed in detail. The paper is so extensive as to require original reading, as an abstract could not do it justice.

Lead and Zinc.

The Daly Judge Mill at Park City, Utah, concentrates an argentiferous lead ore carrying some pyrite and blende. As described in the July 2, 1910, issue of the *Min. Wld.*, by Mr. L. A. Palmer, the system of concentration is briefly as follows: Wet crushing is practiced, the product of crusher and rolls going to a line of screens of 3, 4 and 6-mesh. The oversize from each screen goes to a 3-compartment Hartz jig which delivers both gate and hutch products in the shape of a finished lead concentrate; middling is discharged over the tailboard. The undersize from the last screen goes to a classifier which makes two feeds for fine jigs making finished lead concentrate through the hutch only. The middling from the two coarser jigs is dewatered, screened and treated in Huntington mills, the product from which is combined with the middling from the fine jigs. These are elevated to a dewatering tank, the overflow from which goes to the main settling tank, the settlings being screened and jigged. From the main settling tank, which is composed of 11 compartments overflowing from one to the other, nine Wilfley tables are fed. The tables make five products, lead concentrate, lead middling, zinc middling, tailing and slime. The lead middling is reconcentrated on Wilfley tables where a lead concentrate is recove.ed. Another set of Wilfley tables makes a lead-iron concentrate and a zinc middling. Slime is collected in a settling tank and concentrated on a Wilfley table. The tailings of all the Wilfleys except those treating the middling from the first group of nine, are screened on a 10-mesh trommel, the oversize being rejected and the undersize treated on three Wilfley tables. The zinc middlings are retreated to remove as much of the lead and iron as possible before shipment. Concentration is at the rate of 3.49 tons of crude ore into 1 of shipping product. Extraction is about 90 per cent on lead and gold, and 75 to 80 per cent on zinc and silver. Milling costs are about 95 cents per ton.

The Potter Process, in common with the other flotation processes has suffered from misapprehension as to the exact procedure. One point not clearly understood concerns the use of oil. In the *Mining Magazine* (London) for June, 1910, Mr. W. E. Simpson communicates some interesting facts regarding the operation of the Potter process in Australia during his régime as manager of the Zinc Corporation at Broken Hill. The author states that no oil was used in the Potter process, and is almost certain that Potter never anticipated the use of oil. Neither is it true that oil from exhaust steam probably

got into the solutions, for neither oil nor exhaust steam ever found even a tentative place in the process.

The formation of gelatinous silica by chemical reaction between the quartz and fluorspar in the ore, in the presence of the hot sulphuric acid used, is assigned by Mr. Simpson as one of the elements of success of the Potter process on these ores. The colloid-silica doubtless aided in clotting the sulphides into masses which became buoyant under the action of the heat and the generation of carbonic acid which was continuous. It was proved that if the gelatinous silica were not formed, the process would be so impaired as to render it impracticable.

Another point mentioned by Mr. Simpson, is his discovery that absolute aeration of the material to be treated was necessary before its immersion in the hot acid solution. Prior to aeration the mass has been mixed with the acid solution before the mill water had been drained from the particles of ore and they were coated with a film of mill water. On aerating the mass, the improvement was most marked. In a subsequent 1000-ton test, the actual extraction for the entire time was over 81 per cent of the zinc, and at times the extraction ran as high as 92.5 per cent. Mr. Simpson gives the requirements for the Potter process as follows: That the sulphide particles have clean, air-filmed surfaces, and be not over 1 mm. in diameter; that there must be present, though not naturally added, some gelatinous or oily material for clotting the particles together; and that there must be some means of bringing these clotted masses to the surface, either by the generation of gas or by the buoyant effect of heat or vacuum.

Chemical Engineering.

Protal and Protal-Bakelite.—A paper by Dr. F. G. WIECHMANN before the Am. Inst. Chem. Eng. deals with "a new product for use in the arts." The base of this new material is vegetable-albumin, under which generic term vegetable ivory, the vegetable caseins, glutens, hemi-celluloses, reserve cellulose, horny albumins, etc., are included. The vegetable albumin, from whatever source derived, is treated with one or more substances, which convert it into a new substance, a plastic eminently well adapted for use in the arts and industries. To this new plastic, the name protal has been given. Any and all materials commonly used in the rubber industry may be incorporated with protal. About one hundred different protal compounds have been produced, and, of course, the properties of these compounds vary with the ingredients employed. Protal can be molded, pressed or otherwise formed, into any desired shape. It is odorless, resilient, and can be cut, sawed, filed, polished, tapped and countersunk, like hard rubber. It can be colored by dyes, and all pigments can be incorporated with it. It is non-explosive and is very resistant to the influence of heat and electricity. Among the great number of protal compounds which have been made, there are some which contain rubber, rubber substitutes, shellac, rosins, asbestos, etc. Some of these products remain plastic and moldable for a long time and possess the remarkable quality of hardening on immersion in water. Compounds of protal with rubber, rubber fluxes and some of the so-called rubber substitutes, exhibit a wide range in their properties. They can be made hard, semi-hard or soft, which of these qualities they are to exhibit being determined by the choice of loading materials and by the conditions of heat and pressure governing their production.

Among the most important compounds of protal is protal-bakelite, bakelite being that most interesting and valuable new product of Dr. L. H. Baekeland, which is, as is well known, a condensation product of phenol and formaldehyde. Protal-bakelite possesses many valuable qualities. It exhibits great resistance to nearly all chemical solvents. It is an excellent electric insulator, is capable of taking a high degree of polish, can be produced in almost every color, and is well adapted to the many purposes and uses for which hard rubber and hard rubber compounds are, at present, almost exclusively employed. It possesses the great advantage over hard rubber of not being subject to oxidation, of not softening on the application of heat and of not being attacked by oils and bodies of a similar nature. Any and all materials, organic as well as inorganic, may be incorporated with protal-bakelite, thus giving rise to a great number of compounds which possess very different qualities and properties and which are adapted to a great variety of uses. Protal-bakelite compounds have, as before said, great dielectric strength, ranging from about ten thousand volts to about twenty-six thousand volts per millimeter. It would be impossible to specify all of the uses to which this new plastic may be put. It need only be borne in mind that this is a plastic which can be fashioned into any shape, which can be molded and pressed, which is capable of taking a high polish, which is not affected by water—cold or boiling—which is resistant to practically all chemical solvents, which can be tooled and machined with ease and which is produced in both flexible and rigid form. In cost, this material compares very favorably with rubber and every day witnesses its introduction into new fields of industry. The manufacture of protal and of protal-bakelite is in the hands of a New York concern, the Protal Company, the works being located at Bridgeport, Conn., and at Yonkers, N. Y. (From advance sheets.)

Problems in Chemical Industry.—A paper on this subject was presented by Mr. JOHN T. BAKER, of Phillipsburgh, N. J., before the Am. Inst. Chem. Eng. The number and complexity of the factors involved in chemical operations are so great that many operations are still carried on under the rule of thumb guidance and have not been reduced to a science. On the other hand the trained scientific man is very prone to believe that the matter with which he deals will follow the laws which he has learned, and for this reason he often overlooks valuable facts which the untrained observer sees. The untrained observer ignores laws and systems; tries any suggestion that comes along, and therefore loses much valuable time and labor. The investigator who is successful follows a mean between these paths. A number of practical illustrations of these principles were given. (From advance sheets.)

Electrochemistry.

Ozone.—A paper on the manufacture and industrial applications of ozone was presented by Dr. OSCAR LINDER, of the Western Electric Company, of Chicago. (By a misprint the author's name was given in our last issue, p. 387, as Dr. Snider instead of Dr. Linder. This is herewith corrected.) The properties and methods of formation of ozone were given. The amount present in pure air was given as about 1 part in 1,000,000. The commercial methods of manufacture of ozone are almost exclusively electrical, although ozone is formed by ultra-violet light and heat. The successful operation of the electric ozonizers depends upon the rise of a high voltage (10,000 to 40,000 volts) and dry air maintaining a low temperature. Concentrations of ozone as high as 30 grams of ozone per cubic meter of air corresponding to 2 per cent by volume have been obtained. Yields of 105 to 250 grams of ozone per kilowatt hour have been obtained.

Ozone has been used industrially for a great many purposes. The purification of the air of theatres, schools, restaurants and public halls is quite common in Europe. Ozonizers made especially for ventilating purposes are now on the market and are made either of portable or stationary type. Large installations for water purification are in successful operation. Ozone is well adapted for sterilizing and disinfecting purposes. It is being tested in the quarantine station in New York and at the Pittsburg Homeopathic Hospital. It is believed that ozone will ultimately supplant formaldehyde in disinfecting rooms and buildings. It has also been successful for the preservation of milk, cream, butter, eggs, fruits, meat, etc. Ozone is also an excellent bleaching agent. (From advance sheets.)

RECENT METALLURGICAL AND CHEMICAL PATENTS

Concentration

A New Concentrator designed along entirely new lines is the invention of Mr. Lucien I. Blake, of Boston, Mass., but at present residing in Denver, Col. Briefly stated, the device consists of a flexible endless belt traveling around a system of inclined rollers, with a sufficient amount of sag between the supporting rollers to form a trough, which is the concentrating

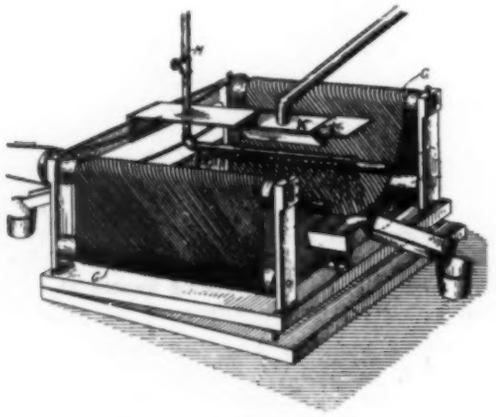


FIG. 1.—CONCENTRATOR.

area. The working surface of the belt is corrugated or riffled and so formed that the riffles run obliquely to the direction of travel of the belt, and in such a direction that the particles of concentrate are carried upward in the trough, while the tailing or reject is allowed to flow to the lower end of the trough. The details of construction may be varied considerably, but the form preferably used by the inventor is that indicated diagrammatically in the accompanying illustration. The flexibile belt *G* may be made of corrugated rubber. Ore in the form of pulp is delivered into a suitable feeding apparatus, indicated by *K*, from which it is fed onto the surface of the belt against the motion of the same. Wash water is introduced through a pipe *M* so located that small jets of water issuing from it will act upon the layer of pulp and aid in the separation of concentrate and tailing. The heavy particles in the pulp will sink into the riffles and travel up the inclined trough, while the lighter particles of gangue will be raised above the riffled surface and be washed off by the wash water. The table thus makes but two products, concentrate and tailing. (964,083, July 12, 1910.)

Smelting.

Blast Furnace Feeding Device.—In the usual operation of a blast furnace such as is used for lead and copper smelting, the gases tend to find paths of least resistance in rising through the column of ore and flux. These paths are usually found along the walls of the shaft. One method of overcoming this tendency to form channels through the smelting mass is to so distribute the charge that the fine matter will lie along the walls and the coarser material in the center of the shaft. This would provide for more even permeation by the gases. Arthur S. Dwight, of San Luis Potosi, Mexico, has patented a device for effecting this mode of feeding mechanically. The device consists of aprons or partitions placed inside the furnace charging doors, and extending both above and below the plane of the feed floor. These vertical aprons are supported in the end walls of the furnace and run parallel with the side walls. In Fig. 2, which is a diagrammatic sectional end view of a furnace, B' is the feed floor, G' the feeding device having vertical side walls g^4 and stays g^7. C' represents the furnace doorway, with sliding door E'. When the ore is charged, the larger pieces of the charge tend to fall toward the center of the shaft upon passing below the apron, while the fine material remains near the side walls. The feeding device presents no obstacle to the usual furnace operations, such as breaking up bridges and crusts, and if the furnace be kept full the amount of cold air which can enter is reduced to a minimum and the size of the flue required to carry away the gases can be reduced as compared with the usual requirements. (959,484, May 31, 1910.)

Reverberatory Furnace.—Various schemes have been proposed to utilize the waste heat from reverberatory furnaces. One of the most successful proposals is the generation of steam in boilers placed at the front of the furnace. The proposal of Frank A. Burns, of Anaconda, Mont., is that reverberatory furnaces be placed in series so that the waste heat from one will aid in effecting the smelting process in the next. He claims a reduced consumption of fuel in proportion to the number of furnaces in the series. The usual amount of fuel would be required in the first furnace in the series, but each succeeding furnace in the battery would require less fuel to be consumed in its fire box. (960,499, June 7, 1910.)

A Regenerative Reverberatory Furnace is the subject of a patent of F. A. Leas, of Oakland, Cal. The idea of the inventor is to provide a reverberatory furnace which can be fired alternately at one end and the other. Oil is the fuel preferred by the inventor of this furnace. From each end of the furnace settling flues extend in any manner desired, but both flues lead to a common short flue, which in turn connects with the stack. In the short flue is a damper of special form, which can be operated to admit air for combustion through the flue leading to one end of the furnace while the products of combustion are passing from the other end out through the stack. When the flue conducting the products of combustion has been heated the operation is reversed by changing the damper, so that the air for combustion now flows through the heated flue, while the flue from the opposite end of the furnace now conducts the products of combustion. Oil burners placed at each end of the furnace are in use alternately, the one in use at any time receiving the preheated air. The furnace thus combines the dust-saving features of the ordinary reverberatory, with the heat economy features of the open hearth steel furnace. At the same time the checker work is eliminated, which is impracticable in a regenerative reverberatory furnace on account of the large amount of dust formed. (962,644, June 28, 1910.)

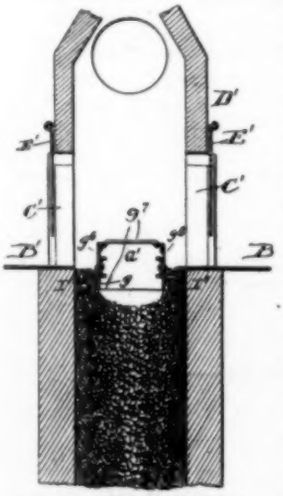


FIG. 2.—BLAST-FURNACE FEEDING DEVICE.

Gold and Silver.

Filtration.—The mechanical treatment of slime pulp is the subject of a number of recent patents, some of which have proved successful, while others are not known to be in use, although their mode of operation is interesting. Edwin Burt, of El Oro, Mexico, has been granted patent on an improved method of filtering slime, in which he prefers to use his apparatus patented some months ago. (931, 267, Aug. 17, 1909). It comprises a rotatable shell in which is placed a filtering medium, on which the filter cake is to be deposited as the shell rotates. The chief difficulty in operating devices of this type has been the lack of coherence in the filter cake during the rotation of the shell, parts of the cake falling off. The object of the present invention is to provide a firm cake. Slime is run into the shell until it is half full. Air or fluid pressure is then gradually generated in the filter shell as filtration pro-

ceeds, until a final pressure of 40 to 50 pounds exists. As the shell rotates at a rate of about 15 revolutions per minute, the liquid portion of the slime is forced out through the filter and a cake forms under pressure sufficient to hold it intact during the necessary subsequent washing. (961,713, June 14, 1910.)

Quite different in form is the apparatus designed by CLYDE H. JAY, of Salt Lake City, Utah. The operation will be understood by reference to Fig. 3. The description of the process appeared prior to the issue of patent papers in the *Western Chemist & Met.*, May, 1909. The inventor's idea is to provide a continuous dewatering, agitating, and filtering process for handling slime pulp. Thickened pulp is directed into the cone-bottomed tank containing filter leaves as shown. The vacuum pump is started and by the time the tank is filled with pulp it has been practically dewatered, and ready to receive the cyanide solution with which it is to be treated. Agitation is accomplished by a Pohle air-lift pump placed vertically in the center of the tank. As soon as it is considered advisable, a vacuum is created in the vacuum tank and filtration of metal-bearing solution commences, which is continued until the solution has been extracted. Wash water or weak solution is then applied and filtered off. The discharge of the pulp is effected by directing a current of water into the filter leaves, causing the cake to drop off, when it is sluiced out at the bottom. The inventor claims low cost of installation for the work accomplished; minimum amount of labor; small amount of equipment for tonnage treated, and maximum extraction in minimum time. (963,710, July 5, 1910.)

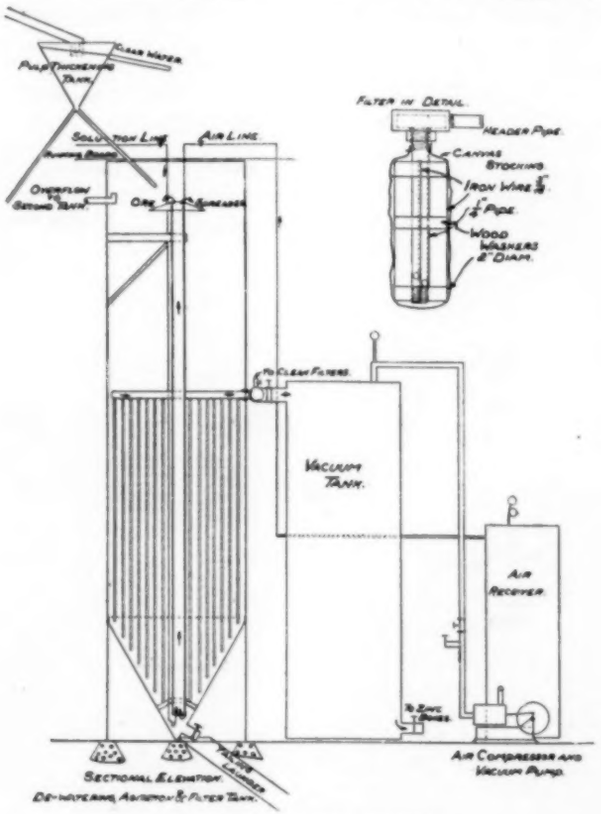


FIG. 3.—JAY CYANIDE PROCESS.

A novel scheme in filtration schemes is that of CLARENCE M. CHAMBERLAIN, of Denver, Colorado, although it does not appear to have advantages over the more common types of apparatus. He proposes to arrange open filter trays horizontally on a steel framework, each filter tray being connected with vacuum apparatus so that slime may be filtered under vacuum. By means of a time wheel fitted with cams, it is arranged to have pulp, weak solution and wash water successively directed into the trays, so that the process of filtration and washing will be continuous. The discharge of the cake is effected by means of scrapers which are so arranged that they may be operated over the surface of the filter trays after the filtration is complete. (959,461, May 31, 1910.)

A better known form of filter, on which patents have been granted to WILBUR A. HENDRYX, formerly of Denver, Colo., but now of New York City, may be used as a vacuum filter or as a decanter as conditions indicate. A round tank with a converging bottom is provided with filter leaves radially arranged around a central well. The filters occupy the upper portion of the tank only, the conical portion being unoccupied. At the bottom of the tank there is horizontal discharge pipe in which a screw propeller may revolve to discharge the thickened or filtered pulp. The flow of pulp is downward through the central well, at the bottom of which is a spreading baffle. When the pulp rises to the height of the filter leaves it may be filtered either by decantation, if clear enough, or by vacuum through the filters. The thickened pulp in the bottom can be discharged through the pipe containing the screw, at the end of which is a gate which can be regulated to allow only a small amount of dewatered pulp to pass. If the filter leaves are used, the cake can be discharged from them by a current of water directed through the inside of the filter, and the pulp discharged through the bottom as before indicated. (960,546 and 960,547, June 7, 1910.)

Precipitation on Aluminium.—The use of this metal to replace zinc as a precipitant for gold and silver in cyanide solutions has been suggested from time to time by several writers, and is used in laboratory schemes for analysis of cyanide solutions. STAFFORD F. KIRKPATRICK, of Kingston, Ontario, Canada, has patented the process for commercial application in an apparatus specially devised for the purpose. The most efficient form in which aluminium could be used for this purpose is as a powder, but powdered aluminium does not easily become wet by water and does not mix with solutions. In order to overcome this difficulty the inventor makes use of an agitating device to engulf the metal in the solution and thus cause it to react with the precious metal cyanide solutions. If the precipitation is carried out in a round tank with a conical bottom, a suitable agitator can be made by attaching propellor blades onto a vertical shaft revolving in the center of the tank. This will produce a vortex into which the aluminium will be drawn and thoroughly mixed with the solution. Sodium or potassium hydroxide is added to the solution, and the inventor gives the following equation as representing the reaction which takes place:

$$6NaAG(CN)_2 + 6NaOH + 2Al = 6Ag + 6NaCN + Al_2O_3 + 3H_2O.$$ (959,757, May 31, 1910.)

Copper.

Smelting Pyritic Ores.—An improvement in the process of smelting copper sulphide ores and other sulphides has been patented by JOHN PARKE CHANNING and FREDERICK J. FALDING, of New York City. They propose to so regulate the supply of air for smelting that practically all of the iron and sulphur available for pyritic smelting shall be oxidized, and that the gases resulting shall be available for the manufacture of sulphuric acid. It is further claimed that a higher grade matte will result from these operations. In the case of pyritic smelting of ores not containing copper, the resulting iron slag will be suitable for the production of pig iron in the blast furnace. The regulation of the amount of air used is to be effected by means of valves near the tuyeres. The top of the furnace is to be so arranged that no excess of air will be admitted to disturb the equilibrium established. If it is found necessary to dilute the gases in order to make them better available for the manufacture of sulphuric acid, the inventors propose a second set of tuyeres above the charge, through which the necessary amount of diluent can be introduced. In regard to the use of carbonaceous fuel, it will be seen that it would be de-

sirable to reduce the quantity as much as possible. The inventors claim that experiments have shown that ½ per cent carbon to each 4 per cent available sulphur in the charge is a proportion which gives gases having sufficient oxygen therein to convert sulphur dioxide into sulphur trioxide. It is further stated that practice has determined that the amount of free oxygen should be considerably more than the amount theoretically required to convert sulphur dioxide into sulphur trioxide. Assuming that the gases contain 6 per cent sulphur dioxide, they would require, theoretically, 3 per cent of free oxygen to convert them into sulphur trioxide. It is found, however, that 7 to 8 per cent free oxygen should be present. The richness of the furnace gases in sulphur dioxide should range between 5 per cent and 7 per cent sulphur dioxide. (963,020, June 28, 1910.)

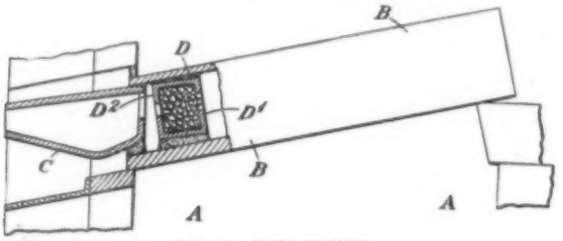


FIG. 4.—ZINC RETORT.

Zinc.

Improved Extraction.—In the ordinary distillation of zinc from its ores it has been found that some sort of mechanical filtration of the metallic fume as it passes from the retort to the condenser improves the quality of the condensed zinc by removing certain impurities, notably lead. Fig. 4 shows the apparatus designed by CHARLES S. BRAND, of Troon, Scotland, for this purpose. *A* indicates the furnace heating area, *B* a retort, and *C* a condenser. The filter container *D* is of fire clay and is filled with material refractory to the temperature employed in the process. A suitable material has been found in pieces of broken clay retorts, silica brick, etc. The container is about 6 in. in length and of a cross-section similar to that of the retort in which it sets. The filtering material is of about 1 cu. in. in size. The ends of the container are perforated to permit the passage of the metallic fume. The apparatus is so arranged that at the completion of the distillation of a charge the container may be withdrawn and the condensed lead poured from it. (963,416, July 5, 1910.)

The Continuous Volatilization of Zinc from its ores is the subject of a patent of HARRY H. HUGHES, of Springfield, Mo. The construction of his furnace provides for the continuous introduction of ore without access of air, for the removal of the molten zinc and the residue, and for the utilization of the carbon monoxide gas generated in the reduction of the ore. Reference to Fig. 5 will make the process plain. The charge of ore to be reduced is fed in at the hoppers 26 and 27, whence it is transferred into the retort 3 by means of a screw conveyor. The retort is encased in a revolving combustion chamber, in which the retort is suitably supported as shown. The retort connects with a condensing extension 8, which further communicates with the portion 11, from which the molten zinc is discharged through 33, and the residue through gates 38 and 40. The combustion chamber is heated by oil burners 32, and provision is further made for the combustion of the carbon monoxide formed in the reduction of the ore, by conducting it into the combustion chamber by means of the pipe 35. The inventor claims that every particle of ore and carbon is brought in contact with the heat quickly and at the proper temperature to effect the production of the maximum quantity of zinc; and further that the condensing apparatus is under such control that the minimum quantity of blue powder will be formed. (13,118, Reissued May 31, 1910.)

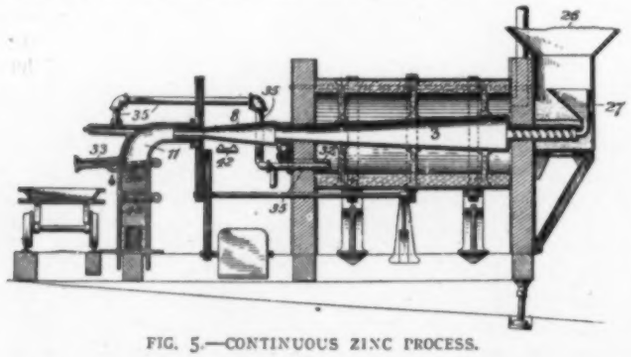


FIG. 5.—CONTINUOUS ZINC PROCESS.

Smelting Lead-Zinc Ores.—For the purpose of cleaning mixed zinc and lead ores containing gold and silver prior to their treatment in the blast furnace, EDUARD DEDOLPH, of Kaslo, British Columbia, proposes to volatilize the zinc and lead as oxides and sulphates, leaving a mixture suitable for the blast furnace. The operation is carried out in a revolving furnace fed with a mixture of the ore and flux which will make a suitable smelting product after the zinc and lead have been volatilized. The fuel may be gas or oil directed into the discharge end of the drum, where a jet of air is also introduced. Two drums can be used in conjunction, the first operating at a temperature of about 800° Celsius, and the second at about 1300 degrees. (959,924, May 31, 1910.)

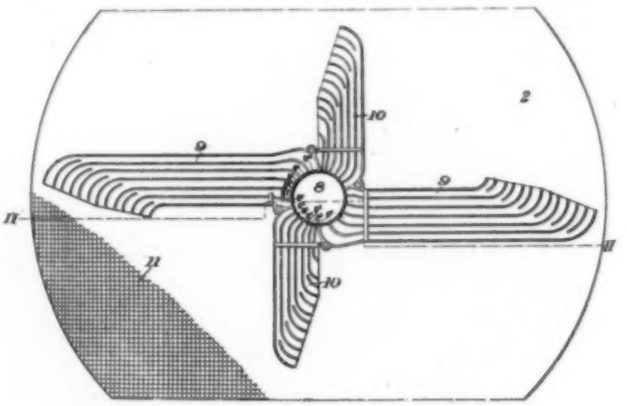


FIG. 6.—GAS SCRUBBER SPRAY.

Chemical Engineering.

Spray for Gas Scrubber.—For the purpose of spraying water or other liquids in gas scrubbers or cooling towers, WILLIAM AHLEN, of Duquesne, Pennsylvania, has patented a device of his invention. In removing finely divided solids from blast furnace gases it is necessary that the sprayed water be uniformly distributed over the cross section of the scrubber. The object of the present invention is to provide such a uniform distribution. Fig. 6 shows a plan of the spray located in the top of a scrubber. It comprises a hollow shaft 8, with rotatable arms 9 and 10. Each of the arms is provided with several passages through which the water is delivered to openings at the ends. Below the spray is a series of screens 11. In operation of the device, water is led into the pipe 8, from which it overflows through outlets into the passages in the arms. The widths of the outlets are proportioned to the lengths of the passages, progressively increasing with the length of the latter. When the water reaches the ends of the troughs or passages it overflows in a plurality of small streams, which fall in a series of annular paths as the arms rotate. The speed is so regulated that no water is thrown from the spray by centrifugal force, and as the numerous small streams fall upon the screens they are broken up into drops so that the effect is that of a fine rain. This offers the maximum cooling and cleansing surface to the rising gases. (961,195, June 14, 1910.)

Micro-Metallograph.

The study of the microstructure of metals with the aid of the microscope and by microphotographs has become in recent years a most important and useful method in metallurgical research and with the aid of improved apparatus enhancing the practical usefulness of microphotography, this method will soon be indispensable in any metallurgical testing or research laboratory.

The micro-metallograph, described in this article, has been developed by the renowned optical works of E. Leitz, Wetzlar, Germany, and embodies various interesting and important omprovements. It is designed for visual observation as well as for the photomicrographic study of metallic surfaces. The construction of the apparatus embodies the principle first applied by Le Chatelier together with certain improvements suggested by Dr. W. Guertler and Dr. Wm. Campbell of the School of Mines, Columbia University, the object being placed upon the stage T, while the objective is underneath. In the act of focusing the stage moves together with the object, whereas the objective and the illuminating system remain stationary. This part of the apparatus is its most important feature (Dr. W. v. Ignatowsky, Zeitschrift f. wiss. Mikroskopie, Vol. XXV, No. 4, 1909). The apparatus has the advantage that it is available with any magnification, besides being readily adjustable in such a manner as to obviate detrimental reflections or glares.

The micro-metallograph consists essentially of three parts, all of which are mounted upon an optical bench, viz., a stand with tube and stage, a lamp with iris diaphragm and illuminating lenses, and a photographic camera.

1. *Stand with tube and stage* (Fig. 1). The stand S consists of an upright which carries the tube together with the illuminating appliances. The stage T may be raised and lowered with the aid of a rack and pinion motion and a micrometric slow motion M.

2. *The lamp.* The lamp A in Fig. 1 may be raised and lowered along the upright S_1 by means of a rack and pinion. The carbons are adjustable axially with the aid of the screws m_1 and m_2. The upright S_1 is made to slide along the bench B_2. The lamp is provided with a carrier S_2 capable of displacement along a slide bar b and fitted with an iris-diaphragm and illuminating lenses.

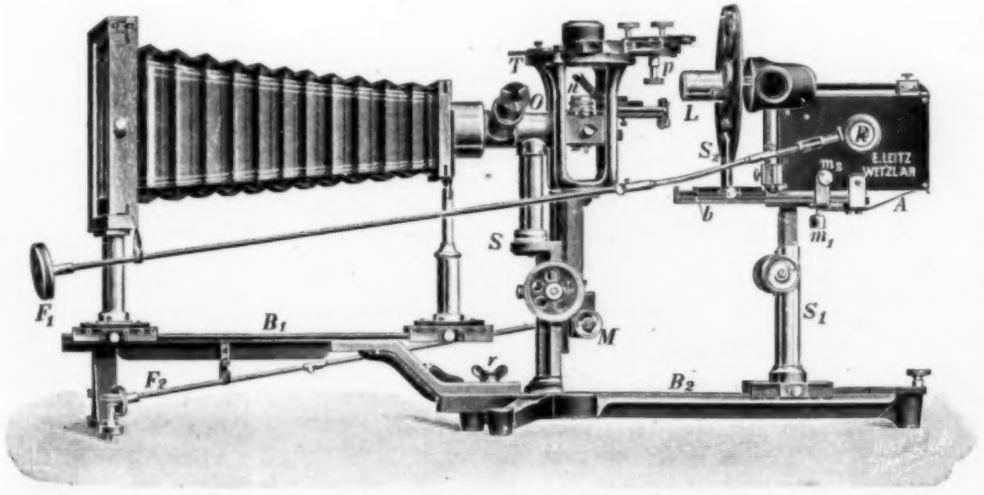


FIG. 1.—MICRO-METALLOGRAPH, STAND WITH TUBE AND STAGE.

The lamp requires an electric current of about 4 amp and with the addition of a suitable resistance may be worked from the ordinary lighting circuit. The lamp is provided with a hood fitted with a dark glass window for watching the state of the carbons. The latter are at right angles to each other. By this arrangement the crater becomes entirely free, which greatly enhances the available amount of light. The arc gap is regulated by means of a hand wheel R or by means of the adjusting key F_1.

3. *The camera.* The camera is ordinarily adapted for 13-cm x 18-cm, or 7-in. x 5-in. plates. The greatest available distance from the focusing screen to the eyepiece is 280 mm. The camera is supported by two uprights, which are capable in displacement along the bench B_1 (Fig. 1). The latter, together with the camera, after untightening the wing-nut r, may be detached from the bench B_2.

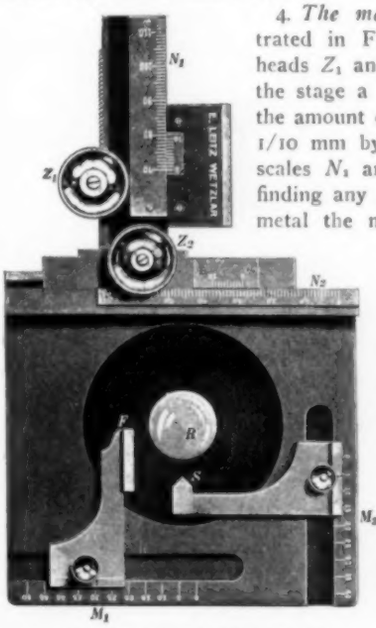


FIG. 2.—MECHANICAL STAGE.

4. *The mechanical stage* is illustrated in Fig. 2. The two milled heads Z_1 and Z_2 serve to impart to the stage a compound cross motion, the amount of which may be read to 1/10 mm by means of two vernier scales N_1 and N_2. As a means of finding any given spot in a piece of metal the mechanical stage is provided with a sliding pointer S, the position of which is registered by a scale M_2. This pointer is placed in contact with a fine file mark on one side of the specimen, while another side rests against the flat F of another sliding arm, the position of which may be seen on the scale. N_1 N_2 and M_1 M_2 furnish accordingly data with the help of which particular points of interest may be recorded and found at any time.

The mechanical stage attaches by means of the clamping screw p (Figs. 1 and 3) to the stage T, after the removal of the stage rings. These rings, of which there are three, fit a corresponding opening in the mechanical stage. In Fig. 2 they are indicated by the letter R.

To operate the instrument for *moderate and high magnification* the lamp A is placed in its lowest position by means of the rack and pinion attached to S_1 (Fig. 1), then moved along the bench B_2 as far away as possible from the stand and the iris-diaphragm I is placed (Fig. 3) in position. The carrier S_2 (Fig. 3) is then moved toward the lamp and the iris-diaphragm closed down to a very small aperture. The screw r (Fig. 3) is loosened and the tube B lowered as far as it will go, after which r is retightened. Finally, the objective side is withdrawn by the knob R_1, and the objective is screwed on. The ocular tube is capable of displacement along its axis and is fitted with a prism, which when the ocular mount is pushed in deflects the rays transmitted through the objective into the ocular. The prism may be put out of action by sliding out the ocular tube so as to allow the rays to pass without obstruction to the focusing screen.

Fig. 4, which shows the path of the rays, will assist the

reader in following the manipulations involved in the adjustment of the apparatus.

Let B mark the position of the source of light, supposing it to be reduced to a point. J represents the iris-diaphragm as shown in Fig. 3. With the aid of the projecting pin-knob s (Fig. 3) the position of the lens L_1 (Fig. 4) is adjusted within

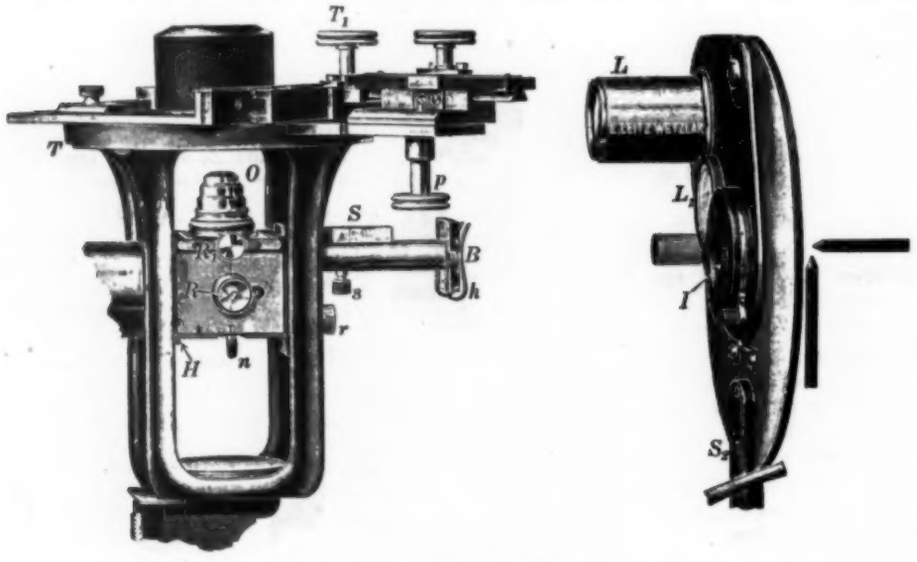


FIG. 3.—DETAILS OF MICRO-METALLOGRAPH.

the tube B (Fig. 3) until an image of the iris-diaphragm is formed at E, i. e., about midway above the upper face of prism P (Fig. 4). The latter may with the aid of the milled knob R (Fig. 3) be turned about an axis at right angles to the plane of the paper and also displaced along AB.

The pencil of rays proceeding from P, after passing through the objective o, illuminates the object, whence that portion which goes to form the final image in the eyepiece or on the ground glass plate is reflected back through the objective, past the prism P and through prism P_1. The latter is likewise movable about an axis at right angles to the surface of the paper and may be adjusted from without by means of the pin n (Fig. 3).

The movable mount of prism P (Fig. 4) affords a means of so directing the rays through the objective o that any light which is reflected by the lens L may be intercepted by prism P to prevent its passage through prism P_1 and the creation of disturbing glares. The width of the pencil emerging from the prism P_1 which is of considerable importance, may be regulated by displacing the lens L_1, which to this end is contained in a movable mount, the distance of L_1 from P being greater with high than with low powers. To direct the correctly illuminated portion of the object into the field of view it is only necessary to turn P_1 about its axis.

Having completed the adjustment of the illumination, the operator may make sure of the absence of glares, which are particularly disturbing in photographic work, by removing the object and replacing it by a non-reflecting body, preferably black velvet. The field as seen through the eyepiece should then be black. After a little practice this test will become superfluous.

Having secured the proper adjustment as controlled through the eyepiece tube, the latter is put out of action and the image is made to appear on the groundglass focusing screen. Photographs are taken by this apparatus with the aid of an eyepiece as well as the objective, a fitting containing an eyepiece being inserted into the sleeve attached to the camera. The final focusing and the adjustment of the carbons is effected with the aid of the transmission gears F_1 and F_2.

The milled head R and the pin n (Fig. 3) are provided with index marks corresponding to convenient initial positions of prisms P and P_1 from which the operator may proceed to make his adjustments.

The image should be evenly bright and undistorted. The presence of distortions should be regarded as an indication that prisms P and P_1 are not properly adjusted.

Tube B (Fig. 3) is fitted with a slide, which should, of course, be opened at the outset. The spring clips at the end of the tube serve for clamping colored glasses or light-filters.

The back of the carrier S_2 (Figs. 1 and 3) is fitted with a shutter for use during photographic exposures.

To operate the instrument for *low magnifications* the lamp should be racked up, as shown in Fig. 1, and moved up as near as possible to S. The glass plate n (Fig. 1) is mounted with its carrier upon the slider S (Fig. 3) and together with tube B raised to the required height and clamped with the aid of screw r. The slide at the end of tube B should be closed. When microsummars (magnifications 9.5 and 13) are used, the operation of the instrument is somewhat different.

For the purpose of cleaning the illuminating apparatus the prisms P and P_1 may be withdrawn from the tube mounted on stand S (Fig. 1). To do so the hook H (Fig. 3) is released, the objective slide is removed, and the illuminating apparatus withdrawn by means of the button R.

When the object is of such dimensions that it cannot be accommodated on the mechanical stage the latter should be removed and the object placed direct upon the stage T (Figs. 1 and 3). The rings of the mechanical stage fit the gap in T. The range of magnifications with fully extended camera is as follows: Without eyepiece and with microsummars, magnifications 9.5 and 13; with eyepiece No. 2 and objectives 1 to 10, magnifications 26, 46, 73, 132, 210, 305, 370, 465, 650 and 710.

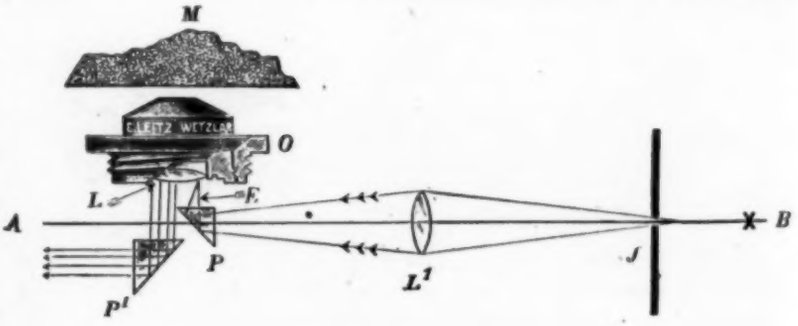


FIG. 4.—PATH OF RAYS.

The optical works of E. Leitz have branches in this country at 30 East Eighteenth Street, New York City, and 1923 Ogden Avenue, Chicago, Ill. This new micro-metallograph has already been introduced in some of the leading metallurgical works in this country, like the Watertown Arsenal, the Scovill Mfg. Co., The Winchester Repeating Arms Co., The American Vanadium Co., etc., and has given complete satisfaction.

High-Speed Forging Press.

A machine which has come into general use very rapidly in the last few years is the high-speed forging press. This machine has been developed abroad and has been extensively used in England and Germany for as much as 10 years, but has been employed in this country, up to a year ago, by only a few of the very large companies who make the heavier class of forgings.

The machine, as shown in Fig. 1, represents the very latest development in forging machines for large work, and Fig. 2 is the same machine, but built with an overhanging frame, which is considered better for the smaller powers. These stand for years of step-by-step progress by Davy Brothers, Limited, of Sheffield, England, who have been designers and builders of—first, steam hammers, and later hydraulic forging presses.

The earlier presses built by this company were of the pump type—that is, the water pressure was derived from a pump which was necessarily slow, with the disadvantage that the water must pass through valves at high pressure, relatively long lines with many joints and numerous drawbacks to a large water system, one being its liability to carry grit which cuts out plungers and packings, another, the liability of damage on account of water hammer.

The next broad step after the direct pump-driven press was the hydraulic intensifier—this made it possible to raise the pressure from somewhere between 1000 lb. and 2000 lb. per

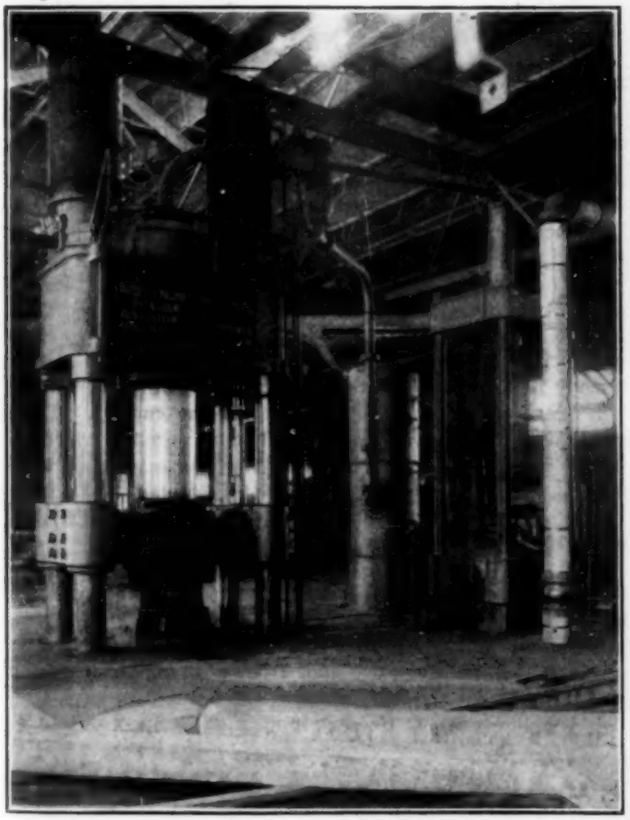


FIG. 1.—FORGING PRESS.

square inch, to 4000 lb. to 6000 lb. per square inch. This makes the main press cylinders and the volume of water correspondingly small, which is a very great advantage in that the press of large power becomes of practical dimensions and the volume of water under this high pressure is relatively small and in an enclosed system, not handled through valves, and only under this high pressure at the instant of delivering the squeeze.

This, of course, was a long step forward, but this machine still required a pump, and the system generally included a large accumulator in addition to the hydraulic intensifier, and was operated by a number of valves and levers. This press was slow, partly owing to the number of valves operated and partly due to the low speed of the hydraulic intensifier.

This intensifier, owing to the fact that the large volume of low-pressure water which must be handled at low speed, was superseded by the steam intensifier. This device took steam direct from the boilers, which acts directly on the low-pressure piston, and gives you at once the high-pressure water required, without the use of the pump. Another great advantage is that

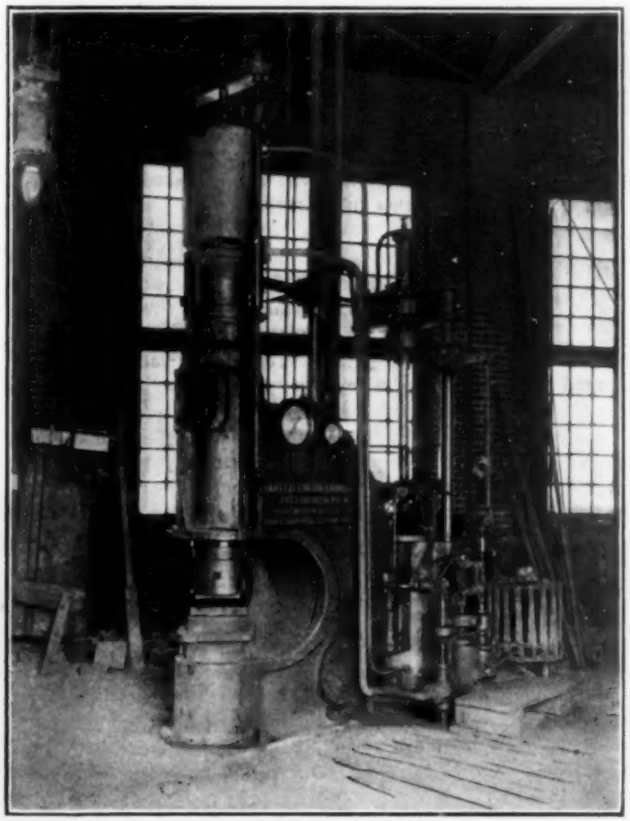


FIG. 2.—FORGING PRESS.

the steam, being highly elastic, can be handled at a very high rate of speed.

There are then two pistons directly connected together, with a ratio of about 40 to 1, and when this device is set vertical, with the water cylinder on top, the return stroke is accomplished by gravity, so that steam is used on one side only. The high-pressure cylinder can then be directly connected to the main cylinder of the press without the water passing through valves. The press head then moves at the same time as the intensifier ram and always a corresponding amount, the intensifier moving about 12 times as far as the press ram. One then has only to control the travel of the intensifier, both as to time and distance, and one has a corresponding control of the presshead, with the advantage that on the presshead the control is as 12 is to 1 better than on the intensifier.

This provides for a quick-working stroke, but it is also necessary to return the presshead quickly. This was accomplished by placing steam cylinders on top of the machine, and also arranged to take steam on one side of the piston only. These cylinders are arranged so that, when forging or intensifier stroke is being used, they do not consume any steam, it simply flowing back and forth in the main pipe.

It now becomes evident that, unless the intensifier be of very large proportions, the stroke of the press will be very short. An intensifier of sufficient capacity to give a practical forging stroke, and also give the proper amount of clearance for a

diversified line of forgings, would be impractical owing to its size, and also on account of the enormous amount of steam required to operate it.

It then becomes apparent that the thing to do is to lower the presshead nearly in contact with the forging, by means of the steam balance cylinders. It is then found necessary to have more or less water in the high-pressure system according to the thickness of forging being operated upon, which, of course, fixes the distance in or out of the main cylinder occu-

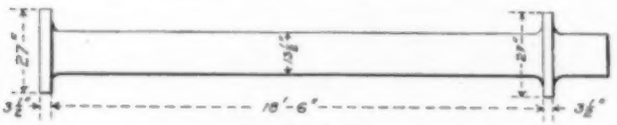


FIG. 3.—SHAFT.

pied by the main ram. This was accomplished by having a tank connected with this high-pressure system and controlled by a check valve. This tank is about half-filled with water, the balance being filled with air at about 30 lb. per square inch. When the presshead is lowered by wholly or partially exhausting the steam from the balance cylinders, the check valve automatically opens, allowing the required amount of water to flow from the tank into the system. This check valve is also arranged to operate with a lever connection so that when it is required to lift the presshead, this can be accomplished by opening the check and allowing the balance cylinders to force back into the tank an amount of water that will give the proper position of presshead.

This action, or long light stroke, takes place always when the intensifier is in its lower position, or at rest, so that it can be readily seen that the presshead can be lowered until it is practically in contact with the forging, and the steam in the intensifier used only for a stroke in proportion to the amount the forging is reduced, and an amount directly in proportion to the resistance met or the work done. This makes a press very efficient in action as, once the presshead is let down on the forging, the low position is maintained until the forging is finished, using no more steam in the balance cylinders as this is forced back and forth from the cylinders to the main steam pipe.

This made a satisfactory press so far as any one stroke was concerned, but when rapid action was required it was found impractical to operate a number of valves or levers for the rea-

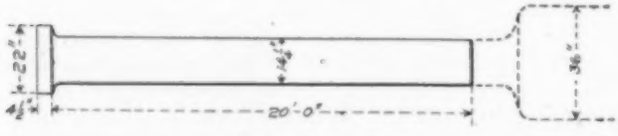


FIG. 4.—SHAFT.

son that if one was moved out of time the result was disastrous to the machine or the work, and it developed slow because it was necessary to move one valve or lever and be sure that the press assumed a corresponding position before moving the next one. This invariably results in the wrecking of the machine through a slip of the operator.

The Davy Brothers, Limited, realizing this, devised the single-lever control, which is patented in the United States as well as abroad. This control operates all valves and permits of putting the press through an entire cycle of motions in one forward and back motion of the lever. The valves cannot be crossed and the press follows exactly the stroke of the hand lever, both as to speed and distance traveled. The lever is so arranged that back of the vertical center line corresponds to the light stroke, which is from 2 ft. to 6 ft., according to the size of the press, and forward of that line is the power stroke from 2½ in. to 6 in. long, according to size of the press.

On account of this arrangement the press can be brought down 2 ft. to 6 ft. on top of a forging and the forging reduced from 2 in. to 6 in. by one forward motion of the hand lever, then the press can be brought all the way back, or any part, by simply moving the lever to the required position, or the lever can be brought back only as far as the center line, which will simply return the presshead the amount of the forging stroke. This forging stroke can then be repeated as often as the lever can be moved. In practice this is as high as 150 per minute in the small presses, down to 60 on the large ones.

This stroke, while it is repeated as often as the blow of a steam hammer, is at a much lower velocity owing to the fact that the press moves only inches where the hammer moves feet. This slow action is very beneficial to the metal because it works it to the center, giving perfectly sound forgings. There is absolutely no danger to the smith owing to the absence of shock or jar. This is very advantageous when shape tools or cutters are used as they can be set up, the presshead brought down on them, and then sighted for position before the pressure is applied.

In cutting or punching, the hand lever can be set for a certain travel and the presshead will not ride forward even though the resistance is suddenly and entirely removed. On account of this, there is no danger from flying tools or work.

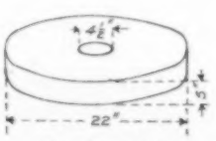


FIG. 5.—DISK.

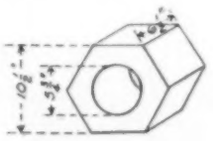


FIG. 6.—NUT.

The machine does not require much foundation, and in working there is no vibration transmitted to surrounding buildings as all strains due to working are taken up in the machine itself.

The United Engineering & Foundry Company, of Pittsburgh, Pa., acquired the sole right to manufacture this machine for the United States, Canada and Mexico about one year ago, and have complete drawings, patterns, jigs, fixtures, etc., for all sizes of these machines, as well as data from the Davy Brothers, Limited, which gives them the advantage of many years' experience abroad on all classes of forgings. They have already built and delivered quite a number of these presses, which are at work in various parts of the country. These presses are working on a diversified line of forgings, which include cogging down ingots, wheel forgings, locomotive forgings, and a general line of forgings.

The high-speed press will do very much more forging than any steam hammer, as can be seen by the following:

Fig. 3.—A 2000-ton press forged the 13½-in shaft, 19 ft. long, with 27-in. diameter flanges, from a 36-in. ingot in two hours, requiring two heats.

Fig. 4.—A 1500-ton press forged a 14½-in. shaft, 20 ft. long, with coupling on one end, from a 36-in. ingot, in one heat, in one hour.

A 1200-ton press on cogging 16-in. x 16-in. ingots to 10-in. x 10-in., and cutting into 18-in. pieces, averaged 16 tons per hour.

Figs. 5 and 6.—A 500-ton press made a disk 22 in. diam-

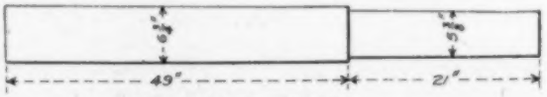


FIG. 7.—SHAFT.

eter, 5 in. thick, upsetting it, from 14 in. square, and punching a 4½-in. hole, as per Fig. 5, in 19 minutes, in one heat. The same size press made and punched a hex nut 10½ in. across flats in 18 minutes, as per Fig. 6, also in one heat.

Fig. 7.—A 300-ton press made a 6¼-in. shaft, 70 in. long, in 12 minutes, from a 7-in. x 7-in. billet in one heat.

Figs. 8 and 9.—A 150-ton press made an eccentric shaft, as

per Fig. 8, from an 8-in. x 8-in. billet, in 13 minutes in one heat. The same press made hex nuts 6½ in. across flats and punched same, upsetting it from 5-in. round stock, in 10 minutes, as per Fig. 9.

The United Engineering & Foundry Company has under-

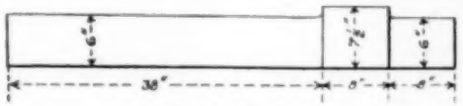


FIG. 8.—SHAFT.

taken to produce a machine in every way first class, among the many refinements of which might be mentioned the following:

The main rams are cast in a chill, which makes them very hard, and are accurately ground to fit the cylinder, which is steel, and bored and lapped out. The intensifier ram is made of hard alloyed steel accurately ground; all joint pins and bushings are hardened and lapped. The material entering into the press is all high grade and nearly all steel. All packings can be removed without dismantling any part of the press, or employing any power outside of the press itself.

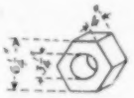


FIG. 9.—NUT.

Jet Machines.

BY OSKAR NAGEL, PH.D.

The working of every jet machine is based upon the fact that a liquid or gaseous jet passes through another liquid or gas (under pressure from a nozzle), causing a motion from the adjoining parts of the mass, so that a vacuum would be formed, if it would not be immediately annihilated by the motion of the particles of the respective substance. The jet draws and sucks in the surrounding mass, mixes with it and transmits to same a part of its own momentum so that both escape together. If now the common velocity is reduced by a counter pressure, a corresponding part of the momentum of the mixture is transformed into pressure which serves to overcome the counter-pressure.

If a steam jet passes through a cooler liquid or if steam is sucked in by a cooler jet formed of a liquid, the steam is condensed in the mixing process and the suction will be the more intense the more complete the condensation.

We now see that in the ordinary jet machine work is produced by using part of the velocity of the passing steam for the transportation of other substances and that in other cases the heat of the steam liberated by condensation is so transformed that it serves for the transportation of other gaseous or liquid substances. In every one of these cases the energy content of the steam is changed. In each case a transformation of energy takes place in the steam.

FIG. 1.—STEAM JET BLOWER.

Since the jet machines on account of their having no moving parts are simpler and more convenient than other machines their application is steadily increasing, especially in the chemical industries. The introductory remarks of this article indicate the most natural and most advantageous uses of the jet machine, namely, to move gases and liquids, the term "move" being used in its widest possible meaning.

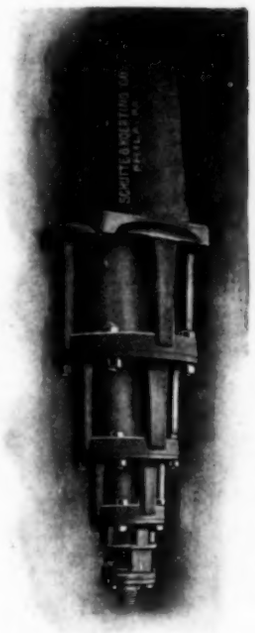


FIG. 2.—CHIMNEY VENTILATOR.

One of the most popular jet machines is the steam-jet blower, Fig. 1, which is mainly used to press air into gas producers or other furnaces and to ventilate cellars, etc.

The chimney ventilators, Fig. 2, which are used to improve the draft of chimneys, also belong to this class, their operation being based upon a series of nozzles in which the motive power of a small steam-jet is utilized for moving the gases; for the removal of acid vapors in chemical work these nozzles are made of lead.

For compressing or exhausting air or other gases, i. e., for vacuum filtration, vacuum distillation, and agitation, the steam-jet exhausters, Fig. 3, are successfully used. (The well-known laboratory type of this instrument is shown in Fig. 4.) A water-jet exhauster for similar purposes is shown in Fig. 5.

Of special interest for the sulphuric acid industries is the lead steam-jet exhauster, and lead blower, Figs. 6 and 7.

The most popular jet apparatus for moving liquids is the injector, which is too well known to require a description at this place. Of greater importance to the chemical industries is the steam-jet syphon, Fig. 8, in which by means of a jet of steam issuing from a nozzle, a vacuum is created which is conveniently utilized for lifting, circulating or transferring liquids. The corresponding apparatus worked by compressed air is called air-jet lift. Its application in the acid industries is easily understood from Fig. 9. The water-jet eductor is built upon the same principle and is mainly used for lifting liquids (Fig. 10).

The above remarks will convey to the reader some idea as to the applications and applicability of jet apparatus. These instruments and their application were developed in Germany by the efforts of the firm of Gebrueder Koerting, Koertingsdorf, near Hanover. In this country these jets are built by the Schutte & Koerting Company, Philadelphia, Pa.

FIG. 3.—STEAM JET EXHAUSTER.

Positive Blowers.

Positive blowers are now being generally introduced in different lines of service for low and moderate pressures up to 8 lb. or 10 lb. per square inch.

The Piqua type of blower has two impellers geared together

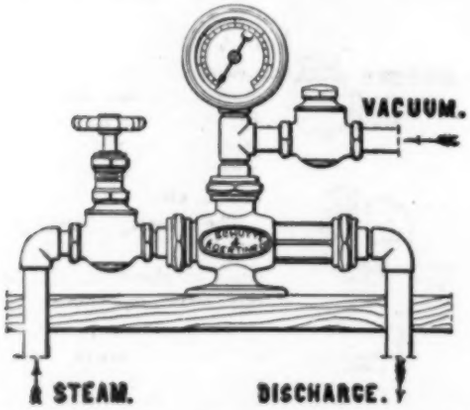


FIG. 4.—LABORATORY TYPE OF STEAM JET EXHAUSTER.

and rotating with very small clearances in rigid semi-cylindrical cast-iron casings. With no internal friction, carefully constructed bearings and flat cut gears dipping in oil, the efficiency of this type of blower is high.

Due to their economy in operation the few parts and simple and rugged construction, and the comparatively low first cost, positive blowers within their pressure limits have a certain advantage over reciprocating air compressors or blowing engines. Also their almost uniform efficiency over the entire range of pressure and volume enables them to readily surpass the centrifugal blower or fan except when this is driven at its exact designed volume and pressure.

For service in the production of grey iron castings, in the foundry cupola, the definite known volume of air delivered, regardless of the changing pressure, makes for an output uniform in quality and increased in quantity.

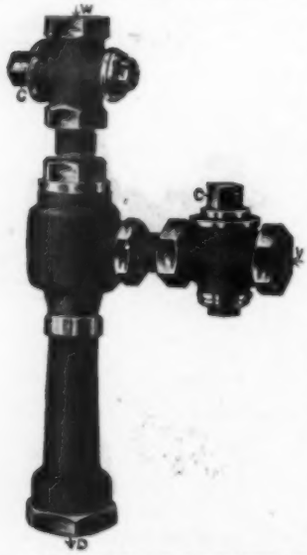


FIG. 5.—WATER JET EXHAUSTER.

In brass and aluminium foundries positive blowers are also giving satisfaction, while for copper and silver-lead smelting the reliability for continuous operation is appreciated.

Blowers for fuel-oil furnaces is another useful application of machines up to 2 lb. and higher pressure blowers have been extensively used to furnish the blast for steel converters.

Among the other uses to which this type of blower is well adapted might be mentioned furnaces for annealing, hardening and tempering; operating sand blast apparatus, agitating, cooling, or evaporating liquids, pneumatic tube conveying, mine ventilation, gas boosters or as exhausters for maintaining low vacuum for gas producers, etc.

The Piqua Blower Company, Piqua, Ohio, has just issued an attractive 48-page catalog M-10 illustrating their line of positive blowers. These run from laboratory size to machines displacing 15,340 cu. ft. of free air per minute and delivering at standard pressures up to 16 oz. or 32 oz. and in the high-pressure class, 3 lb, 4½ lb., 6 lb. and 8 lb. per square inch. The machines are furnished for belt drive or are direct connected to motors or engines.

NOTES

The General Chemical Company of California, with principal office in New York, has been incorporated with a capital of $3,750,000. The directors include William C. Peyton, of San

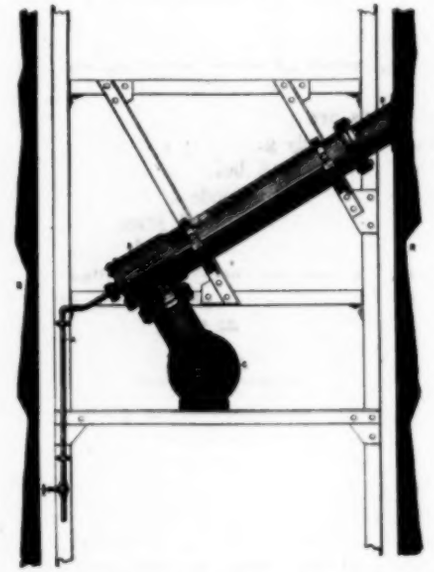


FIG. 6.—STEAM JET EXHAUSTER FOR SULPHURIC ACID PLANTS. (A, STEAM SUPPLY PIPE; B, LEAD EXHAUSTER AND BLOWER; C, GAS SUPPLY; R, CHAMBERS.)

Francisco; Charles W. Millard, Thomas F. Burgess and Frank H. Edmunds, of New York.

The Pierce Amalgamator has been moved from 1654 Champa Street to 1738 Broadway, Denver, Col. This change was made as more roof and better facilities were needed by the company to display their machines, which have been attracting

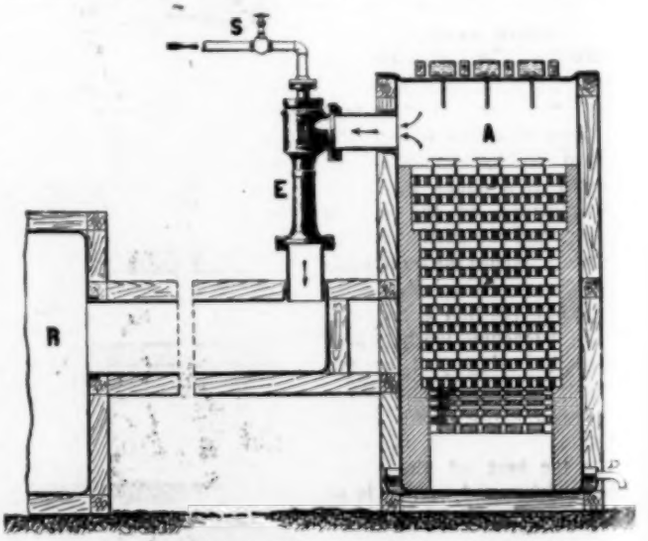


FIG. 7.—LEAD BLOWER FOR SULPHURIC ACID PLANTS.

more and more the attention of metallurgical engineers during the past year. In their new home the company will have spacious office quarters and display room and in connection with this they will have a very complete testing plant in operation ready at all times to demonstrate just what can be accomplished with the Pierce amalgamator.

Recording Instruments.—The Bristol Company of Waterbury, Conn., has just issued "an illustrated index of Bristol's recording instruments for pressure, temperature and electricity, and the William H. Bristol electric pyrometers for high temperatures." This is not a catalog, but a very interesting series of 57 beautifully executed illustrations of the most important Bristol instruments, with a brief and concise statement of their principal features and applications. The illustrations include recording pressure gages, vacuum gages, draught gages, differential pressure gages, water level gages, recording thermometers and pyrometers, recording voltmeters, ammeters and wattmeters, time recorders, recording tachometers, etc.

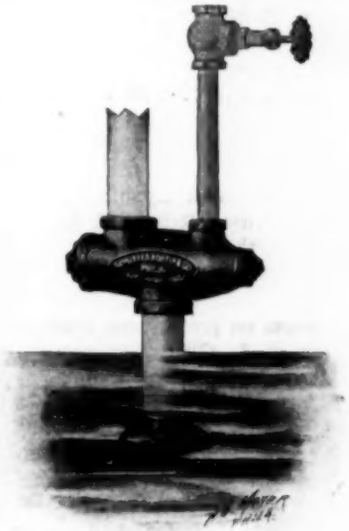


FIG. 8.—STEAM-JET SYPHON.

Sulitelma Mines.—Advice has been received from the Sulitelma Company's mines in Norway that for the month ending May 30 763 tons of concentrates, containing 6.22 per cent copper, were produced by the Elmore process.

The **Brown Instrument Company,** of Philadelphia, has just issued a 45-page illustrated catalog on Brown electric pyrometers, both indicating and recording pyrometers of the thermo-electric type. In the second part the Brown fixed-focus radiation pyrometer is described. The pamphlet contains numerous interesting and instructive illustrations and discusses in detail the methods of installing the various parts of the electric pyrometer and its use in different industries.

The **Williams Patent Crusher & Pulverizer Company,** of St. Louis, Mo., has just issued a new cement edition of their catalog describing their various well-known crushing and grinding machines, with special reference to their use in the cement industry.

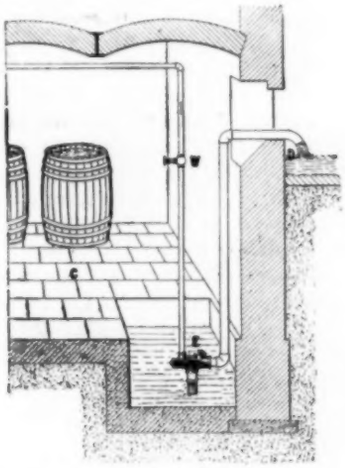


FIG. 10.—WATER JET EDUCTOR FOR LIFTING LIQUIDS.

The **Taylor Instrument Companies** called their branch managers and principal salesmen together to their annual convention in Rochester, N. Y., on July 11. Three busy days were spent by the officers of the companies and the men in reviewing the general conditions in the thermometer industry and outlining new measures and policies for the coming year. The Taylor Instrument Companies are just closing their most successful year. Although the present plant of the companies was occupied only in 1906, the growth of the corporation has been so rapid as to necessitate material additions for the manufacture and handling of "Tycos" products. One building, measuring 100 ft. x 160 ft., will be used exclusively as a wood-working shop for the preparation of all the lumber entering into the products of these companies. A dry kiln is also under construction for the drying of rough lumber. These buildings, with a new wing added to the main building, will make an addition of more than 40,000 sq. ft. of floor space to the present plant.

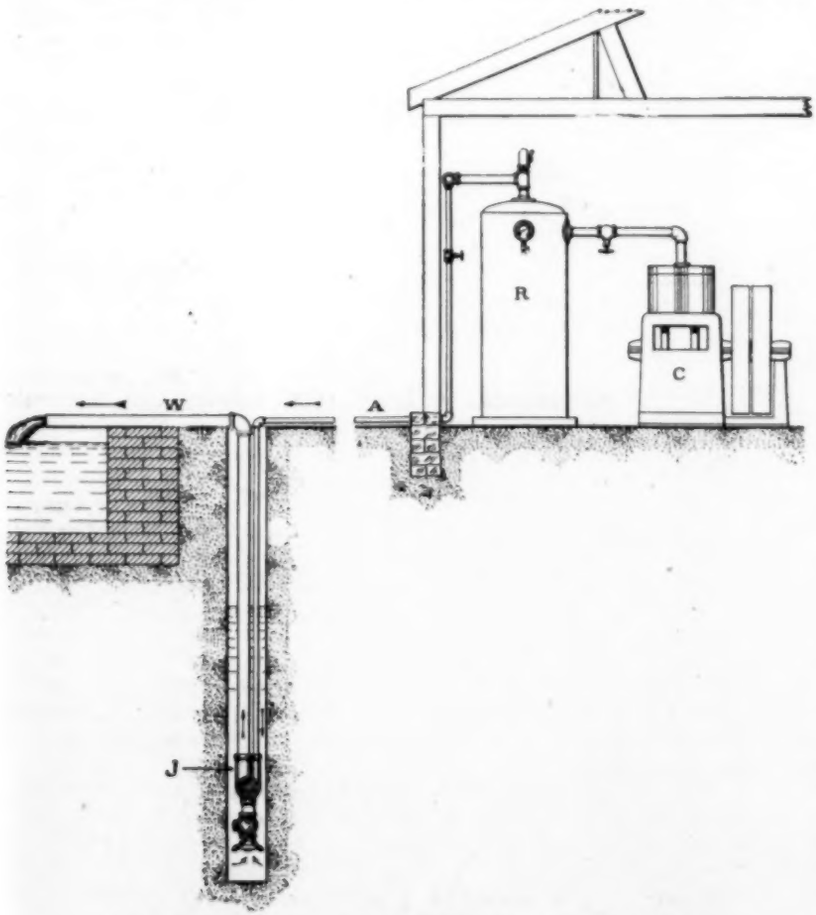


FIG. 9.—APPLICATION OF STEAM-JET SYPHON IN ACID PLANT.

Turbine Fan.—A booklet on a new turbine fan has been issued recently by J. R. Robinson, of Pittsburgh, Pa. This fan is of "mixed-flow" type, i.e., air is taken in axially at the center, turned through 90 deg. and discharged radially by centrifugal force at the circumference. The vanes are curved to pass the air through with minimum friction, and it is claimed that considerable saving in power is effected over that required by various standard fans on the market. The fan is suitable for mine ventilation or for ventilation of power houses and other buildings and for high-pressure work with cupolas, forges, smel-

ters and refineries, furnaces, glass houses, etc. Listed sizes run from 18 in. diameter, 17 in. width, 2500 r.p.m. and 21,000 cu. ft. per minute discharge to 312 in. diameter, 105 in. width, 120 r.p.m. and 1,200,000 cu. ft. per minute approximate capacity. A number of these fans are in satisfactory service, particularly in mines, and in addition to a power saving they require a comparatively small amount of space and have proved themselves capable of maintaining an unsually high water gage. In a later issue we expect to give further information with illustrations and data.

Why Solder Perishes.—Under this caption we find in a British contemporary some notes which appear to be based on recent publications in *Zeit. f. Phys. Chemie.* (Vol. 68, 1909, p. 214 and vol. 71, p. 301). V. Haslinger observed some time ago that all the seams soldered with tin of a tinned iron can were broken, because the tin had crystallized. By contact with this "sick" tin other polished tin surfaces could be infected so that they would get rough. E. Cohen has recently investigated this phenomenon and confirmed Haslinger's observations. The explanation is not that the tin passes from one crystalline form into another, nor has the phenomenon to do with the so-called "tin pest" in which ordinary tin changes to gray tin, which occurs below 20° C. In our case higher temperatures favor the crystallization of the tin. What really happens is that "machined tin" is in a forced or metastable state; tinned iron sheets, rolled tin (tin foil), and any brilliantly polished tin are in this metastable state; at higher temperatures the tin gives up its metastable state and crystallizes and the continuity of the soldered seam is broken. E. Cohen and J. K. Inonye have recently extended this investigation to other metals. We intend to deal with this phenomenon which is also of practical interest, in greater detail in a future issue.

Oil and Gas Burning Furnaces and Appliances.—Some interesting circulars have been issued within the past few months by Tate, Jones & Company, Inc., of Pittsburgh, Pa. No. 129 covers fuel-oil burning equipment for open-hearth furnaces and illustrates the Kirkwood oil burner, swinging and water-cooled types, also a reversing valve stand and the oil pumping, heating and regulating system which goes along with the installation. Oil is being used successfully as a fuel in many open-hearth steel furnaces; it is generally the cheapest fuel except in the immediate neighborhood of coal and natural gas fields, and the heat of the furnace is more easily regulated than is possible with producer gas. Even where natural gas and coal are abundant it has been found advisable in many instances to install auxiliary oil-burning equipment for emergency use. The Kirkwood burner is a combination high and low-pressure oil burner in which the oil is fed to the burner at from 15 lb. to 25 lb. pressure, and atomized by a small quantity of compressed air or steam at a slightly lower pressure. An air blast of from 2 oz. to 6 oz. is used to supply the necessary amount of oxygen for complete combustion. This burner is suitable for open-hearth furnaces, glass-house furnaces and leers, small steam boilers, brick kilns, rivet forges, heating and annealing furnaces, etc.

PERSONAL

Mr. **J. F. Callbreath, Jr.,** secretary of the American Mining Congress, has been to Los Angeles, making preparations for the meeting of the Congress there Sept. 26 to Oct. 1.

The wedding is announced of Mr. **James M. Camp,** chief chemist of the Carnegie Steel Company, and Miss Isabella R. Wightman, of Pittsburgh, Pa., on June 30.

Mr. **Louis S. Cates** has accepted the position of superintendent of mines for the Ray Consolidated Copper Company, Ray, Ariz.

Mr. **Robert Kennedy Duncan,** who has been professor of industrial chemistry at the University of Kansas, will hold a similar position in the University of Pittsburg the coming year.

Mr. **C. E. Foster,** who has been connected with the Taylor Instruments Companies of Rochester, N. Y., has finished the special work for which he came to this country and has now returned to England.

Mr. **H. W. Hardinge,** president of the Hardinge Conical Mill Company, sailed on July 16 on a professional trip to Sweden.

Mr. **H. E. T. Haultain,** of Toronto, has been appointed professor of mining in Toronto University.

Mr. **E. N. Hawkins** will spend the summer in Denver, after an absence of several years, during which time he has been engaged in professional work in the Northwest.

Mr. **George T. Holloway,** consulting metallurgist, chemist and assayer, announces the removal of his office and laboratories from Chancery Lane to 9-13 Emmett Street, Limehouse, London E.

Mr. **Frederick Lyon,** assistant managing director of the U. S. Smelting, Refining & Mining Company, was in Salt Lake City recently after a trip to the company's smelter at Kennett, Cal.

Mr. **Frederick H. Morley,** of Denver, has gone to British Columbia on professional business.

Mr. **E. L. Oliver** has opened an office at Oakland, Cal., and will engage in consulting metallurgical practice. He has resigned his position with the North Star Mines Company, at Grass Valley, where he developed the Oliver continuous slime filter.

Dr. **Harrison Patten,** of the Bureau of Soils, Washington, is the author of a novel, "The Town and the Trust," just brought out by the Neale Publishing Company.

Prof. **Robert Peele,** of the Columbia School of Mines, New York, has gone to South Africa on a six months' leave of absence to study mining operations.

Dr. **F. W. Traphagen,** professor of metallurgy in the Colorado School of Mines, is spending the summer in Montana in professional work.

Mr. **George J. Young,** of the Mackay School of Mines, Reno, Nev., recently returned after an eight months' leave of absence, during which he visited many European institutions of mining education.

BOOK REVIEWS.

Transactions of the American Institute of Chemical Engineers. Vol. II.: 1909. 308 pages; illustrated. Price, $6. New York: D. Van Nostrand Company.

Besides official notices and reports, the second volume of the *Transactions* contains the following papers: Charles F. McKenna, "The Centering of Great Industries in the New York Metropolitan District"; Charles E. Munroe, "Chemical Industries of America"; Samuel P. Sadtler, "Conservation and the Chemical Engineer"; William D. Ennis, "Efficiency Limits in the Power-Gas Producer"; O. K. Zwingenberger, "Utilization of Low-Grade Fuels in the United States"; A. Bement, "Heat Efficiency of Smokeless Combustion and Heat-Absorbing Capacity of Boilers"; A. Bement, "Chemical Composition of Illinois Coal and the Relation of Heating Value to Composition"; Samuel P. Sadtler, "Creosote Oil from Water-Gas Tar"; John C. Olsen, John S. Weiffenback and John R. Brierly, "Experiments on the Case-Hardening of Steel by Gases"; R. K. Meade, "Automatic Acid Egg"; John G. Dean, "Method of Clay Control for the Manufacture of Cement"; Jerome Alexander, "Colloids and the Ultramicroscope"; Stephen P. Sharples, "Utilization of Waste India Rubber"; Launcelot W. Andrews, "Permanence and Acidity of Hydrogen Peroxide Solutions"; Wm. M. Booth, "Commercial Extraction of Grease and Oils"; F. J. Wood, "Glycerine Refining in Multiple-Effect Stills"; A. C. Langmuir, "Advantages of the Multiple-Effect Distillation of Glycerine and Other Products"; Edward R. Taylor, "Electric Furnace for the Smelting of Iron Ore"; Edward R. Taylor, "Waterways and Water-Power."

The frontispiece is an excellent portrait of the president, Dr. Charles F. McKenna.